

FUTURE TECHNOLOGIES FOR ENERGY-EFFICIENT IRON AND STEEL MAKING

Jeroen de Beer

ECOFYS, NL 3503 RK Utrecht, Netherlands; e-mail: J.debeer@ecofys.nl

Ernst Worrell

Lawrence Berkeley National Laboratory, Berkeley, California 94720; e-mail:
e_wged@dante.lbl.gov

Kornelis Blok

Department of Science, Technology, and Society, Utrecht University, Padualaan 14,
NL-3584 CH Utrecht, Netherlands; e-mail: k.blok@chem.uu.nl

KEY WORDS: energy efficiency, manufacturing industry, technology assessment, exergy
analysis, smelting reduction, near net shape casting

ABSTRACT

Techniques for the reduction of the specific energy consumption for iron and steel making are identified and characterized to assess the potential for future energy-efficiency improvement and research and development priorities. World-wide average specific energy consumption for steel making is estimated to be 24 GJ/tonne. The most energy-efficient process requires 19 GJ/tonne for primary steel and 7 GJ/tonne for secondary steel. Seven specific smelting reduction processes and four groups of near-net-shape casting techniques are described and evaluated. In the longer term, the specific energy consumption for making steel from iron ore can be reduced to 12.5 GJ of primary steel per tonne. A further reduction of up to 2.5 GJ of crude steel per tonne may be achieved when techniques are developed that can recover and apply heat from the hot steel at a high temperature. The specific energy consumption for secondary steel making can be reduced to 3.5 GJ/tonne by energy-efficient melting and shaping techniques.

CONTENTS

1. INTRODUCTION	124
2. PAST TECHNOLOGICAL DEVELOPMENT OF IRON AND STEEL PRODUCTION	126
2.1 <i>History of Iron Making</i>	126
2.2 <i>History of Steel Making</i>	128
2.3 <i>The Current Situation</i>	131
3. ENERGY SERVICE AND THEORETICAL SPECIFIC ENERGY CONSUMPTION	134
3.1 <i>Description of the Energy Service</i>	134
3.2 <i>Calculation of the Theoretically Lowest Energy Demand</i>	134
3.3 <i>Heating and Melting of Iron</i>	137
3.4 <i>Iron Ore Reduction in the Blast Furnace</i>	137
3.5 <i>Comparison with Practical Processes</i>	140
4. EXERGY ANALYSIS OF AN INTEGRATED STEEL PLANT	140
4.1 <i>The Reference Plant</i>	141
4.2 <i>Results of the Exergy Analysis</i>	146
4.3 <i>Conclusions</i>	150
5. IDENTIFICATION AND SELECTION OF LONG-TERM ENERGY-EFFICIENT TECHNIQUES	150
5.1 <i>Gathering of Information</i>	150
5.2 <i>Selection of Energy-Efficient Techniques</i>	151
6. CHARACTERIZATION OF LONG-TERM ENERGY-EFFICIENT TECHNIQUES ...	152
6.1 <i>Smelting Reduction Processes</i>	153
6.2 <i>Near-Net-Shape Casting</i>	173
6.3 <i>Scrap-Based Process</i>	181
6.4 <i>Steel Making at Lower Temperatures</i>	186
6.5 <i>Waste Heat Recovery at High Temperatures</i>	188
6.6 <i>Conclusions Concerning the Potential of Long-Term Energy-Efficiency Improvement</i>	192
7. DISCUSSION	197
8. CONCLUSIONS AND RECOMMENDATIONS	199

1. INTRODUCTION

The iron and steel industry is the largest energy-consuming manufacturing industry in the world. In 1990, its global energy consumption was estimated to be 18–19 exajoule (EJ), or 10–15% of the annual industrial energy consumption (1). Figure 1 shows that annual world steel production has increased from about 100 million tonnes in 1945 to about 770 million tonnes in 1990 (2, 3). Global steel production is expected to grow further, by about 1.7% a year, mainly because of an increase in steel consumption in developing countries (1, 4). The apparent steel consumption per capita in these countries is only one seventh of that in Organization for Economic Cooperation and Development (OECD) countries, but this situation is likely to change (1). Whereas the crude steel production in OECD countries has remained fairly stable at 320–370 million tonnes per year since 1980, the production in developing countries is growing steadily at a rate of more than 6% annually and reached about 240 million tonnes in 1993 (1). This growth is expected to continue. As a result, global

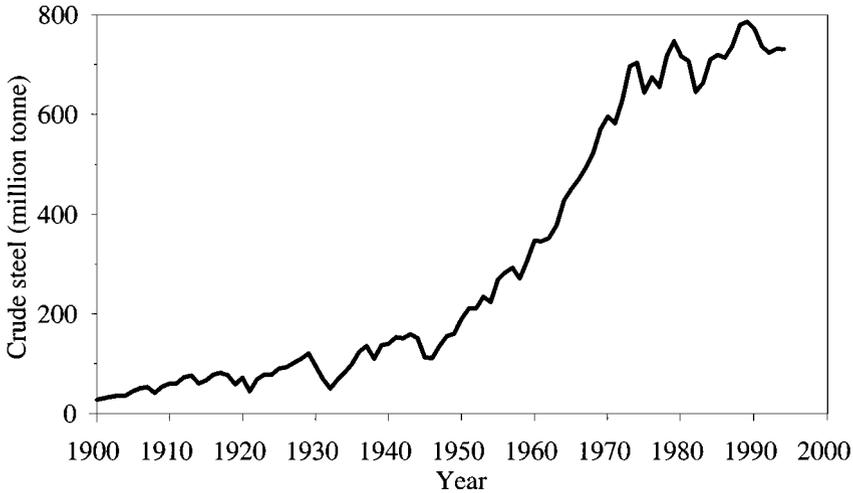


Figure 1 Development of world crude steel production from 1900 to 1994. Data up to 1991 are taken from Reference 2. Data for 1992 are taken from Reference 33.

steel production might rise to 1280 million tonnes in 2020, assuming a business-as-usual scenario. In this scenario the global energy consumption of the iron and steel industry is projected to increase to more than 25 EJ in 2020 (1).

Improvement in the energy efficiency of steel production is one option to counteract the increasing demand for energy. There have been many studies of the potential for energy-efficiency improvement that can be realized in the short term, i.e. in less than 10–15 years from now (see e.g. 5–9). There have also been some estimates of the energy demand of the steel industry in the longer term. For instance, in a report of the World Energy Council it was estimated that on the basis of an advanced technology scenario, primary energy demand would grow to about 20 EJ in 2020 (1). This amount would be a 20% reduction in the energy demand projected by the aforementioned business-as-usual scenario. Although scenario studies may give us some insight into possible developments, they usually give little information about the techniques required to bring about the energy-efficiency improvements. More information is needed on each technique, and the information needs to be collected and presented in a systematic way. Only then will it be possible to assess the associated research and development (R&D) requirements and to determine how much a specific technique will contribute to an improvement in energy efficiency in the longer term.

The objective of this paper is to identify and characterize, through a systematic approach, techniques that can contribute to an increase in the energy efficiency of steel making, to estimate the long-term potential for energy-efficiency improvement, and to assess R&D priorities. This approach has been described

extensively in a previous energy-efficiency study that focuses on the paper and board industry (10). It consists of three steps: First, an energy analysis of the current process is performed. Second, an inventory is made of techniques that might contribute to an improvement in the energy efficiency in the long term. Third, each technique is characterized by determining the impact on the energy demand and costs of the production process, by evaluating the current state of development, and by assessing the technical change required to bring the technology to commercialization.

In this paper, the historical perspective of iron and steel making processes is described. Next, an analysis is made of the theoretically lowest amount of energy required to produce iron and steel. In the following section, an exergy analysis is made of the currently prevailing steel production route, the blast furnace–basic oxygen furnace route. On the basis of the results of the energy and exergy analyses, a description is given of possible routes for energy-efficiency improvement. Next, different techniques are described that can improve the energy efficiency of steel making. The potential impact and costs of each technique are evaluated. Finally, the methodology applied and the results are discussed, and conclusions are drawn. In addition, recommendations for policy makers are given.

2. PAST TECHNOLOGICAL DEVELOPMENT OF IRON AND STEEL PRODUCTION

In this section a brief history of the major iron- and steel-making processes is presented. The aim is to place these processes in a historical perspective and to describe energy-efficiency improvement in the past. We first discuss the main processes involved in the making of pig iron, which is reduced iron ore that still contains impurities, mainly carbon. Then we deal with the main processes used to improve the quality by removing impurities, with an emphasis on steel-making processes.

2.1 *History of Iron Making*

The first record of the use of iron goes back to 2500–2000 BC (9). It is believed that in that period iron was not produced deliberately but was obtained from natural resources, e.g. meteorites (11). Deliberate production of iron began in about 1300 BC with the use of charcoal as fuel and reducer, in small furnaces that made use of cold air. Evidence for the existence of such furnaces has been found in Africa, Asia, and central Europe (11, 12). The temperature that could be achieved in these furnaces was probably below the melting point of iron. The product had to be hammered for it to be freed from slag and to make wrought iron. When better blowing devices were introduced, the temperature could be raised, and liquid, high-carbon iron was formed. In 1300 AD the Stuckoven

was introduced in Germany. Although the Stuckoven was only 3–5 m high and 1–1.2 m in diameter (11), its design was essentially the same as that of the modern blast furnace. Charcoal was used as fuel. Based on data on the use of charcoal to produce pig iron and bar steel in the United Kingdom in the period 1540–1760 (13, 14), we can make an estimate of the reduction in the energy demand in this period. The charcoal consumption to make pig iron decreased from 5.5 to 2 loads of charcoal per tonne of pig iron in this period. This is an improvement in energy efficiency of about 0.5% a year. [At that time, charcoal was delivered in cartloads to the ironworks. A load did not seem to have a standard measure. Hammersley (14) gives a range of 13.5–17.5 hundredweight (cwt) (1 cwt is about 50 kg) for a load of charcoal. Assuming an average lower heating value of 29.5 GJ/tonne (14a), 1 load of charcoal equals 20–26 GJ.] Pig iron was converted to bar steel in the finery process. Between 1540 and 1760, the energy demand for the finery process decreased from 16 to 4 loads per tonne of bar steel, or a decrease of 0.6% a year (14). Because both the demand for charcoal used for steel making and the amount of pig iron needed per tonne of steel decreased, the overall energy-efficiency improvement is greater than 0.6% a year (14).

Because of the weak structure of charcoal and the height of the blast furnace, the capacity of blast furnaces was limited. This is because the coal in the mix forms the supporting structure of the furnace charge. Coke is much stronger and does not have this disadvantage. Coke was first used around 1718, but its application in the United Kingdom remained limited to one site until the 1750s (15). Before 1750, charcoal was cheaper than coke, but this situation changed in the period 1750–1790. In addition, the amount of coke required for pig iron reduction decreased markedly in this period (15). In 1750, coke pig iron made up 5% of the total UK pig iron production; by 1790 it made up 90% (15). The development of the average coke consumption for pig iron production from the time when coke-fired blast furnaces were introduced is shown in Figure 2 (3, 16). Three main periods of energy-efficiency improvement can be distinguished. First, in the period of the first diffusion of the process, between 1760 and 1800, a reduction in coke demand of almost 2% a year was achieved, mainly by the introduction of steam engines, which permitted the use of higher blast pressures (16). Second, in the nineteenth century, demand for coke declined further, by an average of 1% a year. The use of regenerators to preheat the blast accounted for much of this reduction. Finally, in the period 1950–1990, reduction of demand for coke was 3.4% a year on average. This reduction in demand was achieved by, for instance, increasing the iron content of the ore, using ore agglomeration, raising the temperature of the hot blast, and the use of blast furnaces with a larger inner volume. On average an improvement in the energy efficiency of iron making of 1.4% a year was achieved in the period 1760–1990.

Up to the 1960s the blast furnace was the main process for reducing iron oxide. Direct reduction processes have been in use since ancient times but

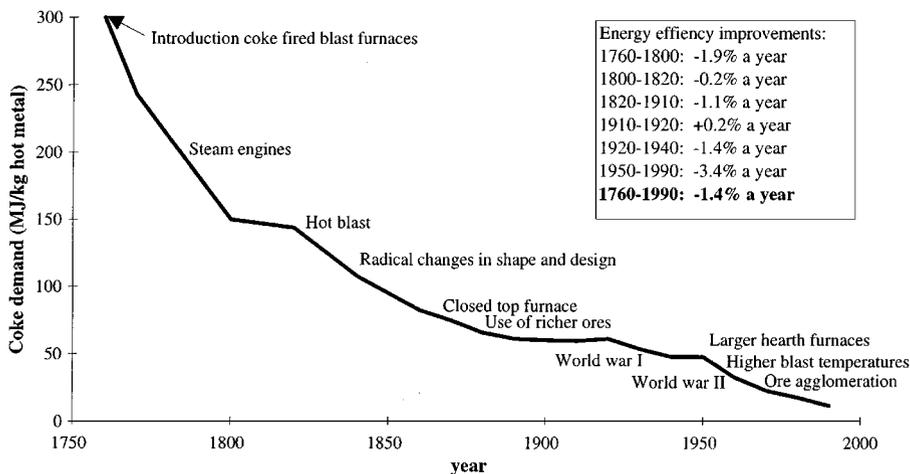


Figure 2 Development of the coke demand for pig iron making (3, 14). MJ, megajoule.

gained renewed interest in the 1960s. Several direct reduction processes have been developed and are now in use. From a more recent date are the smelting reduction processes, which are still under development. These two processes are discussed later in this paper.

2.2 History of Steel Making

Table 1 gives an overview of the history of processes to upgrade the quality of pig iron and steel-making processes. Three lines of development are distinguished: (a) refining processes, (b) (re)melting processes, and (c) processes that both refine and melt. In refining processes, carbon and other impurities present in pig iron, like silicon and manganese, are removed. In this line of development we also consider processes that free the pig iron from slag. The product is not steel but is, for instance, wrought iron. Melting processes are processes whereby the steel is melted only to be cast. Because no refining takes place, the composition of the feed should equal the composition of the desired product.

The oldest process for *refining* iron is the inefficient, charcoal-fueled finery process, which was widely used in the eleventh and twelfth centuries (17). The product was wrought iron rather than steel. At the end of the eighteenth century, the puddle process was introduced. First it was used to make iron, but around 1850 it was converted to make steel by refining pig iron on the hearth of a reverberatory furnace. The product of a puddle process was a semifluid steel, which had to be forged. However, the technique had limited success.

In 1855 Bessemer obtained a patent on a new process, at present known as the Bessemer converter. In 1860 the first Bessemer process went into operation

Table 1 History of steel-making processes^a

Year	Refining processes	Refining and melting processes	Melting processes
	Finery (from 12th century) ^b Puddling (1785) ^b		Crucible (18th century) ^b
1850	Bessemer (1860)	Open hearth furnace (1864)	
	Thomas (1878)		
1900			Electric arc furnace (EAF) (1900)
1950		Basic oxygen furnace (1952)	
		Oxygen bottom blowing (1967)	UHP-EAF (1970) DC-EAF (1985)
2000			

^aBased on References 62 and 17. Approximate dates of first industrial application are shown in parentheses.

^bData from Reference 17.

(17, 18). The principle of the Bessemer converter is still followed: The oxidation of carbon and other impurities provides enough heat to melt the metal. In the Bessemer converter, cold air was blown from the bottom through a refractory-lined vessel. In theory, no additional fuel was required. In practice, about 1 tonne of coal per tonne of steel was consumed (18), equal to about 30 GJ/tonne of steel. Other advantages were reduced refining time and investment costs. However, there were several disadvantages: It was impossible to remove sulphur and phosphorus; the product became brittle after some time because of the large quantities of nitrogen dissolved in the steel; and the process of oxidation was so fast that it was difficult to control the product quality (18).

In 1878 an adapted version of the Bessemer process, the Thomas process, was introduced. This process allowed the production of low-phosphorus steel from high-phosphorus pig iron. The Thomas process used a basic refractory lining instead of the acid refractory lining of the Bessemer process; it is therefore also called the basic Bessemer process (17).

In the meantime the open hearth furnace (OHF) (or the Siemens-Martin furnace) was developed in France. In an OHF, pig iron and scrap are melted on a hearth of a reverberatory furnace by the heat of a flame. The OHF resembles the puddle process; the difference is that in the OHF, air and gaseous fuel are preheated by heat exchanging with the combustion gases in what was called a regenerative gas furnace (19). With the regenerative gas furnace it was possible to attain temperatures sufficiently high to melt steel. The process had two main advantages over the Bessemer process: (a) Pig iron and scrap of any

composition could be melted and (b) good control of the steel quality was possible. The price paid for these advantages was higher investment, higher energy consumption, and longer refining time.

In 1952 another new process for steel making was introduced: the basic oxygen furnace (BOF). The process is also known as the Linz-Donawitz (LD) process, named after the two cities where the Austrian steel company VOEST built the first two BOFs. A BOF is an improved version of the Bessemer process. Oxygen is blown through a water-cooled lance from the top into the converter. The advantages of using pure oxygen instead of air are that the gas volume to be heated and compressed is smaller, no nitrogen can dissolve in the metal, and the heat generated by the oxidation of impurities is greater and adequate to melt 20–30% additional scrap. The BOF had far better energy efficiency than the OHF, and refining was 10 times faster. The idea of using oxygen was already mentioned by Bessemer in 1856. Two factors impeded earlier implementation. First, industrial methods for producing large quantities of oxygen became available only around 1950. Second, experiments were initially directed at blowing oxygen from the bottom into the converter. This configuration generated so much heat that the tubes through which the oxygen was blown (tuyeres) could withstand only one single heat (17). In the 1970s several processes were developed that used the concept of bottom-blowing. Currently, combined blowing processes, i.e. processes that combine the advantages of top- and bottom-blowing, are in operation at some sites. There are also totally bottom-injected processes, e.g. Q-BOP. The state-of-the-art process is a basic oxygen furnace that uses top gas recovery and additional scrap melting. Modern BOFs are net energy producers.

The oldest *melting* process is the crucible process. A closed pot with an average capacity of 25–35 kg—the crucible—was filled with solid wrought iron and heated in a shaft furnace (17). The process required a charge with a composition close to that of the product (17) and about 7 tonnes of coke per tonne of steel (18).

A completely different route to steel is the melting of iron in a bath at a high temperature achieved with the help of electric arcs: the electric arc furnace (EAF). First introduced in the late nineteenth century, its application was limited to specialty steels (18). At present EAFs are used to produce a whole range of products. EAF technology is flexible with respect to inputs. All types of iron can be handled, as can 100% scrap. Furthermore, it can be built separately from blast furnaces and coke ovens. Performance of EAFs has improved tremendously, as is illustrated in Figure 3. The figure shows that in the period 1965–1990, electricity demand declined from 630 to 350 kWh/tonne of steel (2.3% a year on average) and electrode consumption declined from 6.5 to 2.2 kg/tonne (4.2% a year on average). So that a comparison can be made, both consumptions are

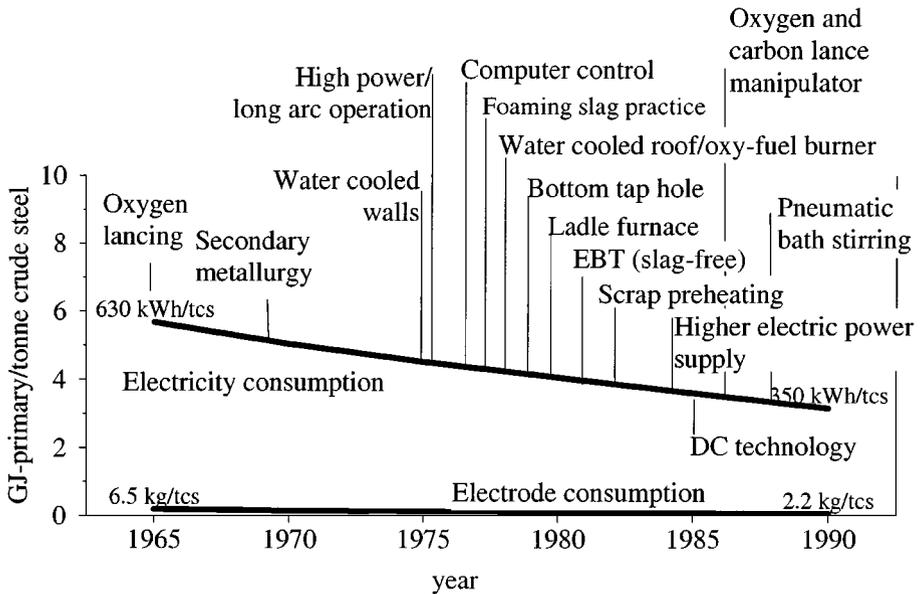


Figure 3 Development in the energy use of electric arc furnaces. Based on data from References 76 and 108. GJ, gigajoule; tcs, tonne of crude steel.

recalculated in terms of primary energy consumption (in gigajoules) per tonne of liquid steel. [It is assumed that the 1 kg of electrode material equals 30 megajoules (MJ) and that electricity is generated with an efficiency of 40%.] Nowadays, refining also takes place in the EAF with the help of oxygen being blown into the furnace.

2.3 The Current Situation

Figure 4 shows the proportion of different iron and steel production processes in the world production of iron and steel. The blast furnace is the most widely used production process for iron. The basic oxygen furnace is still the main steel production technology, but the proportion of the electric arc furnace is increasing steadily. Three steel production routes are illustrated schematically in Figure 5. This figure also gives specific energy consumptions (SECs), expressed in gigajoules of primary energy per tonne of crude steel (GJ/tcs), for the different production routes (20). The SECs represent best-practice values, i.e. the lowest values actually achieved in one plant. As can be seen from the figure, the SEC differs considerably depending on the process route. Even large differences in SEC occur with the same production method. The SEC for an integrated

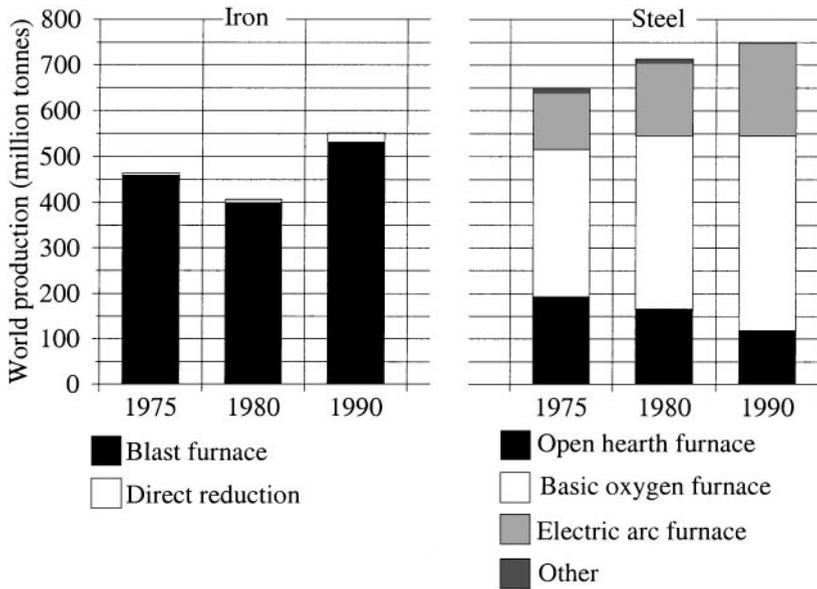
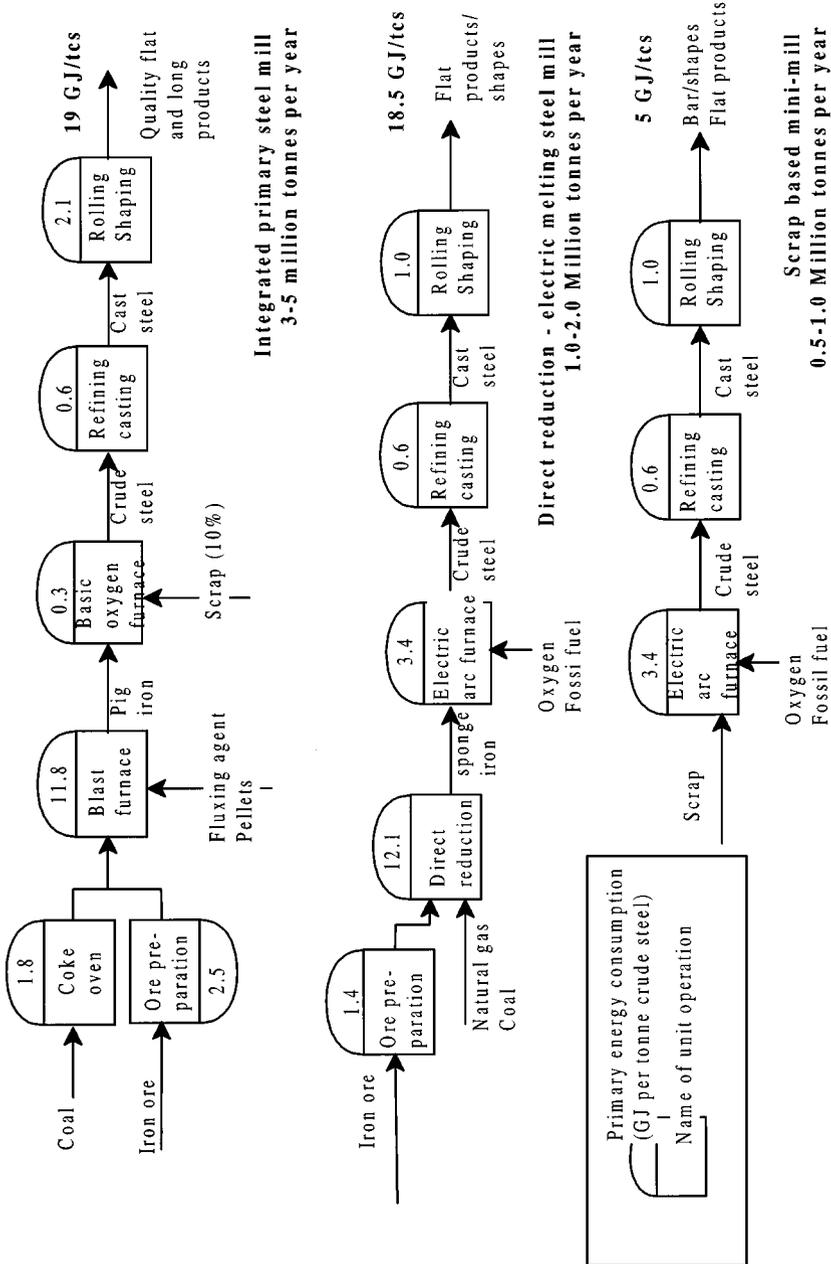


Figure 4 Production volumes of the main iron and steel production processes in 1975, 1980, and 1990. Data for 1975 and 1980, except on direct reduction, are taken from P Norris, International Iron and Steel Institute (unpublished data). Data on direct reduction for 1975 and 1980 are taken from Reference 95. All 1990 data are taken from Reference 2.

primary steel mill varies from 19 to 40 GJ/tcs (1). The direct reduction–electric arc furnace (DR-EAF) production route shows less variation in SEC because the technology is newer and there are far fewer plants in operation. EAF steel making itself has become far more efficient over the past 25 years, as we have shown. Old EAF plants have a SEC that is considerably higher than that given in Figure 5. Worldwide average SEC for steel making in 1990 is estimated to



Figure 5 Flow sheets of contemporary iron- and steel-making processes. The specific energy consumption (SEC) per unit of operation is also shown, expressed in GJ of primary energy per tonne of crude steel. The SEC of the total processes represents the most efficient plants. Data for the primary steel mill are taken from Reference 5. It is assumed that the input of the basic oxygen furnace consists of 10% scrap. All other data are taken from Reference 20. The data for direct reduction are based on the Midrex process. The input of the electric arc furnace in the second production process consists solely of direct reduced iron. All processes end with a hot rolling mill. A cold rolling mill and other finishing operations are not taken into consideration because of the large variation per product. The typical annual capacity is also given. GJ, gigajoule; tcs, tonne of crude steel.



be on the order of 24 GJ/tcs (1). [World crude steel production for 1990 is estimated at 771 million tonnes and the primary energy demand at 18.6 EJ (1).]

3. ENERGY SERVICE AND THEORETICAL SPECIFIC ENERGY CONSUMPTION

The aim of this section is to determine the theoretical specific energy consumption (SEC) for making iron. We start by describing the energy service and thereby set the boundaries for analysis. Thereafter, we determine the theoretically lowest SEC required to perform this energy service. Finally, we consider the theoretically lowest SEC for two important ways of producing steel, i.e. melting of scrap and reduction of iron ore in the blast furnace.

3.1 *Description of the Energy Service*

An energy service is defined as the objective of energy use (21). Energy services can be defined at different levels. The level of definition affects the scope of energy-efficiency improvements. Consider the following energy services: (a) making a material with certain well-described properties, such as strength and resistance etc; (b) making steel, without any further specification; (c) making steel from iron ore.

Each indicated energy service can be used for describing the production of steel. However, the scope of the energy-efficient alternatives differs considerably. In the first case, the production of materials that can compete with steel are taken into consideration, e.g. strong synthetic fibers competing with steel cables. In the second case, scrap recycling and melting are an important option. In the last case, only processes that start with the reduction of iron ore are taken into account.

Although substitution by other materials is an important option for improving the energy efficiency of society, this option is not considered here because the focus of the paper is the energy-efficiency improvement of processes. [For studies of the improvement of material efficiency, see Worrell (22).] In this study we use the second description of the energy service. Thus, recycling of scrap is taken into consideration. The production of steel according to the blast furnace–basic oxygen route is taken as the reference process, because this process is the main production route for steel.

3.2 *Calculation of the Theoretically Lowest Energy Demand*

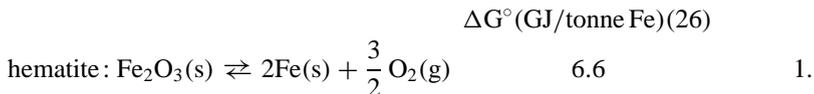
The theoretically lowest energy demand is the amount of energy required to perform the selected energy service without taking into account practical processes. The theoretical steps required for the production of steel from ore are

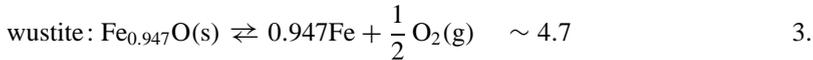
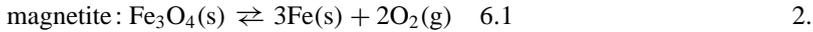
(a) separation of iron oxide from other compounds in the ore, (b) reduction of iron ore, (c) adjusting the composition to make the desired steel, and (d) shaping the steel in the form of the product. When steel is made from scrap, the theoretical steps are (a) upgrading the scrap and (b) shaping the steel in the form of the desired product. The energy required for mining and transporting the ore and for recycling the scrap are not taken into consideration. We give a brief explanation of each step.

PRODUCTION OF STEEL FROM IRON ORE Many minerals in the earth crust contain iron. Besides iron, these minerals, or ores, can contain many other compounds, mainly other oxides, e.g. SiO_2 , Al_2O_3 , and MnO_2 . The iron content of iron ore can be as low as 30%, but it is usually in the 60–70% range (23). Oxides are the most important iron ores. There are three types of iron oxides: hematite (Fe_2O_3), magnetite (Fe_3O_4), and wustite (Fe_{1-y}O , with $y = 0.045-0.135$) (23). Hematite is the most abundant oxide of iron.

1. Separation of iron ore from other compounds in the ore In homogeneous oxides there is a three-dimensional network of covalent bonds. There is strong ionic and covalent bonding. Breaking this bonding requires high energy input, which is reflected by relatively high melting points (24). In multicomponent solids, such as ores, the entropy of mixing should also be taken into account. The entropy of two components that are mixed is smaller than the entropy of the separate components together. Mixing the compounds results in a decrease in the entropy and thus an increase in Gibbs free energy by an amount that is equal to the temperature times the entropy of mixing. For iron ore, consisting of 77% hematite, 15% SiO_2 , 5% magnetite, and 3% other compounds (25), the difference in Gibbs free energy compared to the separate components is calculated to be -0.04 to -0.08 GJ/tonne Fe. It is assumed that all iron in the ore can be recovered. To separate the mixture into the individual compounds, the same amount of energy has to be supplied to the mixture. There are two reasons for regarding this amount of energy as an upper limit for the minimum energy required to recover iron compounds from the ore. First, energy demand is based on the separation of the mixture into its pure compounds, whereas we are interested only in iron oxide. Second, ideal mixing is assumed, while in practice compounds will appear in clusters in the ore. In these clusters no mixing, or less mixing, occurs between different compounds; thus the entropy of mixing is smaller.

2. Reduction of iron oxide Pure oxides can be decomposed into elements, according to the following reactions:





We base the theoretically lowest SEC for the reduction of iron ore on the Gibbs free energy for reaction 1 because hematite is the most abundant iron oxide.

3. Adjusting the composition of the iron to make the desired steel Steel consists mainly of iron. Besides the elements derived from the ore and coke, mainly C and some Si, Mn, P, and S, other elements are added to make alloy steels, including Zn, Cr, Cu, Ni, and Mo (23). Adding these elements does not require energy. The production of these additives may require a significant amount of energy. Because this amount varies, depending on the type and amount of additive, we do not take it into account. New compounds can be formed by a reaction of the elements and compounds present during cooling or heat treatment, e.g. iron carbide and ferromanganese. As an indication for the energy required for these reactions, we consider the formation of iron carbide. The Gibbs free energy of formation of iron carbide (Fe_3C) is 0.11 GJ/tonne of Fe_3C (26). Because the average value for the carbon content in steel is less than 0.5% by weight, the maximum theoretical energy demand for iron carbide formation is 0.002 GJ/tcs.

4. Shaping the steel into the form of the desired product Finally, the steel is shaped into the desired form, and the surface can be adjusted to give the steel certain properties. The difference in the energy content of shaped and nonshaped steel is small. Also, in theory, the changes in the surface properties require hardly any energy.

We can conclude that the theoretical SEC for making steel from ore equals that of one step: the reduction of iron ore. The theoretical energy demand for the other steps is less than 1% of that for the reduction of iron ore. The energy for iron ore reduction is liberated when iron returns to the more stable iron oxide, a process known as rusting. Unfortunately, this energy is hard to recover. In practice, the energy demand for crushing and grinding, pelletizing, and/or sintering iron ore, along with shaping, may constitute a considerable part of the energy demand for making a steel product.

PRODUCTION OF STEEL FROM RECYCLED SCRAP Steel scrap is recycled from many sources. The quality of the scrap depends on the source. One of the largest sources for recycled steel is from automobile bodies and frames. This scrap contains large amounts of zinc, which was used as a surface layer. If not removed, the zinc negatively affects the quality of the steel.

1. Upgrading scrap The quality of scrap is not uniform. It is possible to recycle homogeneous, relatively pure scrap. However, as steel is increasingly

being used in combination with other materials, or is being coated, a major part of the scrap resource will be contaminated with other metals like zinc, nickel, copper, and tin and with polymers and other materials. If we assume that there are no covalent or ionic bonds, the minimum amount of energy that has to be supplied to the mixture to obtain the pure components equals the entropy of mixing times the temperature. We assume that this amount of energy is on the same order of magnitude as that required for the separation of ore into its components, thus less than a maximum of 0.1 GJ/tonne of iron.

2. Shaping of the steel into the form of the desired product For shaping steel, the same conclusion can be drawn as for shaping of primary steel: The theoretically lowest energy demand for this process is negligible.

We can conclude that, in theory, making steel out of scrap requires hardly any energy. However, practical processes require more energy than in the theoretical cases discussed above. Particularly, ore preparation and shaping of steel, of which we neglected the energy demand in this theoretical discussion, will contribute to a higher energy demand. In the next sections we discuss the theoretically lowest energy demand for melting iron and the chemical conversion that take place in a blast furnace.

3.3 *Heating and Melting of Iron*

Figure 6 shows the heat demand for heating and melting pure iron. When pure iron is heated, the lattice structure changes three times. Each change requires the input of transition energy. There are four forms of pure iron, known as α , β , γ , and δ , with transition points at 760°, 907°, and 1400°C. Each transition has its own transition enthalpy. Fe- δ melts at a temperature of 1535°C. Heating iron from 25° to 1535°C and subsequent melting requires 1.36 GJ/tonne of Fe. Of this amount, the total enthalpy demand for all transitions is 0.35 GJ/tonne of Fe. The melting requires the largest part of this: 0.29 GJ/tonne. The melting point of iron is lowered when carbon is dissolved in the iron. When the carbon content is 4.3%, a typical value for pig iron, the melting point is lowered to 1150°C. The enthalpy demand for heating iron from 25°C to the melting point is reduced by about 0.3 GJ/tonne by this temperature decrease. Heating and melting of pig iron theoretically requires 1.05 GJ/tonne; melting of steel, which has a low carbon content, is close to 1.36 GJ/tonne. When the iron cools to environmental temperature, this energy is released again.

3.4 *Iron Ore Reduction in the Blast Furnace*

Iron in oxides has a positive oxidation state and therefore must gain electrons to become free iron. This result can be achieved in several ways, for instance, chemically—a chemical reductant provides electrons—or electrochemically—a direct current provides the electrons. In many metallurgical processes, high

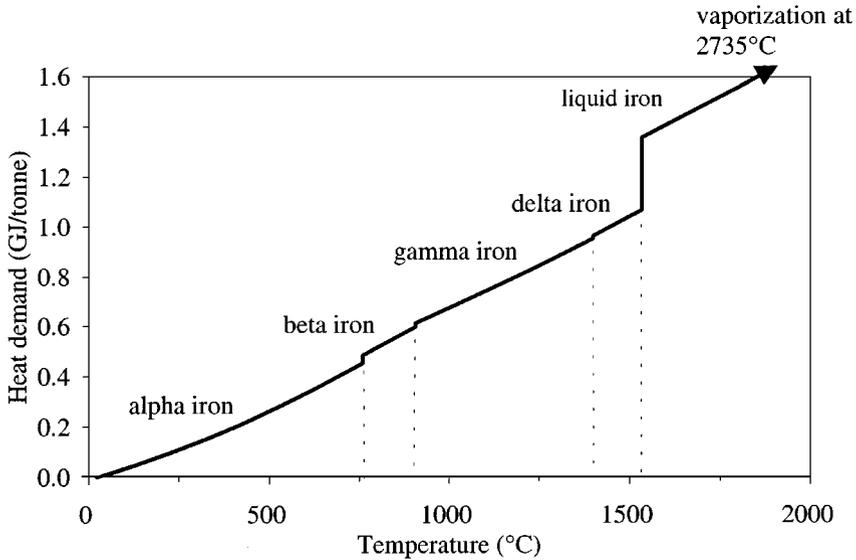
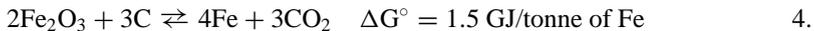


Figure 6 Heat demand for heating and melting pure iron (data taken from Reference 26). GJ, gigajoule.

temperatures are used to promote reactions kinetics and to shift thermodynamic equilibria. A combination is also possible. Aluminum production using the Hall-Herault process, for instance, is a combination of both routes, performed at high temperature.

Iron ore is reduced in the blast furnace with a chemical reductant, carbon (actually carbon monoxide) at high temperatures. Theoretically, this reaction can be described as follows:



The calorific value of pure carbon is 32.8 MJ/kg (26). According to reaction 4, 161 kg of carbon is required to produce 1 tonne of iron, equal to 5.3 GJ. The total minimum energy demand for the reaction is therefore 6.8 GJ/tonne of Fe. Note that this amount is only slightly higher than the theoretical minimum energy demand of Reaction 3. A temperature of more than 900°C is required to let Reaction 4 proceed thermodynamically. In practice, the set of reactions that take place in a blast furnace is far more complex. A short description of these reactions is given below. Carbon is largely converted to CO, which is the main reducing agent in the blast furnace.

REACTIONS THAT TAKE PLACE IN THE BLAST FURNACE Iron ore, coke, and limestone are added to the blast furnace from the top. At the bottom, hot compressed air is blown into the furnace. The reduction of iron ore takes place in two stages: (a) the gasification of carbon and (b) the reduction of ore by carbon monoxide. The main reactions that occur in a blast furnace are as follows:



The temperature zones indicate the zones that can be found in a blast furnace; the hottest zone is at the bottom. Coke is gasified at the bottom (e.g. Reactions 11 and 12), providing the heat and the high temperature required for some reactions. Hot gases ascend, and carbon dioxide can react with coke according to the Boudouard reaction (10) to form more carbon monoxide. The temperature of the gas decreases rapidly because heat is exchanged with the coke bed and with molten materials coming down, and because of the endothermic Boudouard reaction and the direct reduction of molten iron oxide. Direct reduction of FeO with carbon (9) occurs only when FeO is in the liquid phase. The melting point of FeO is 1370°C. Carbon monoxide rises in the furnace, reacting with wustite (8), magnetite (7), and hematite (5). At lower temperatures, the Boudouard reaction proceeds in the opposite direction (6). Molten iron trickles down and collects in a well at the base of the furnace. Although the melting point of iron is 1530°C, a pasty, porous mass is already formed at 1200°C; this is related to the fact that carbon is dissolved. Impurities are removed by reaction with calcium oxide, and a slag is formed. The molten slag floats on the molten iron. Silica that does not react with calcium oxide is reduced by carbon, increasing the energy consumption.

3.5 *Comparison with Practical Processes*

When the theoretically lowest SEC is compared with the SEC of practical processes, the following conclusions can be drawn.

1. The minimum SEC for making a steel product from iron ore equals the energy demand for iron ore reduction that is 6.6 GJ/tonne of steel. In modern blast furnaces, carbon is supplied in the form of coke, coal, and sometimes fuel oil. The total carbon demand is in the range of 350–400 kg of carbon/kg of pig iron (3). Additional energy is supplied by the hot blast. Furthermore, energy is recovered with the blast furnace gas. The net SEC of a modern blast furnace is in the range of 12.5–15 GJ/tonne of pig iron (3). This figure is about twice the theoretically lowest SEC of Reaction 4 and also of Reaction 1. In theory, the SEC for pig iron reduction can be reduced by about 50%. The SEC of modern integrated steel plants, including all other processes, is three times as high. Consequently, the theoretical potential for improvement of the SEC for steel making from iron ore is 65%.

2. The minimum SEC for making a steel product from scrap is negligible. Scrap is melted in modern EAFs with a final energy input of about 1.5 GJ/tonne (3.5 GJ/tonne on a primary energy basis). In theory, the potential for reduction of the SEC is 100%, when the minimum SEC for making steel from scrap is used as the reference. Note that the value of 1.5 GJ/tonne is about 10% above the energy required for heating and melting steel (the composition of scrap is almost similar to that of steel).

4. EXERGY ANALYSIS OF AN INTEGRATED STEEL PLANT

In this section we perform an exergy analysis of an integrated primary steel plant (see Figure 5 for the unit operations of such a plant) to locate the main exergy losses in the process and evaluate their cause. Exergy is the amount of work obtainable when some matter is brought to a state of thermodynamic equilibrium with the common components of the natural surroundings by means of a reversible process (27). It is comparable to Gibbs free energy; the difference is that the common compounds in the environment are taken as reference rather than the elements.

Exergy analyses of selected processes in an integrated steel mill have been described in the literature. For example, Bisio & Poggi present exergy analyses of the sinter plant (28), thermal energy recovery from semifinished products and by-products (29), and blast furnace top gas pressure recovery (29). Bisio performed exergy analyses of scrap remelting (30) and to investigate the opportunities for recovery of heat from molten slag (31). Szargut et al give

exergy analyses of the blast furnace, the BOF plant, walking beam furnaces, and an open hearth furnace (27). Stepanov gives an analysis of a complete integrated steel mill (32). However, Stepanov's mill differs considerably from modern steel mills. For example, it includes an open hearth furnace instead of a BOF plant. To our knowledge there has not yet been published an exergy analysis of a complete modern integrated steel mill.

Because we needed information on the location and the cause of exergy losses, we conducted such an exergy analysis ourselves. Before we present the results of our analysis, we describe the reference plant that we used for the analysis and discuss whether this plant is representative of other integrated primary steel plants.

4.1 *The Reference Plant*

The exergy analysis is based on a hypothetical reference plant described by the International Iron and Steel Institute (IISI) (8). The data for this plant were compiled by a group of international experts on energy use in the iron and steel industry and are based on actual operation data from plants in many countries. The plant is made up of components that were considered to be the most energy-efficient techniques at that time (early 1980s); they were technically proven and commercially viable. The specifications of the main unit operations of this reference plant are given in Table 2.

To assess whether use of this plant as the reference for our analysis is justified, we compare the reference plant with modern integrated steel mills.

1. The IISI reference plant has a net primary energy consumption of 19.2 GJ/tcs (8). Figure 7 compares this value with average SECs for primary steel making in several countries. The IISI reference plant, although designed in the early 1980s, is still fairly efficient compared with the practices in most countries.
2. The IISI reference plant does not use fuel injection into the blast furnace. Nowadays, coke is partially replaced by fuel (coal or oil) injected through the tuyeres into the blast furnace. Fuel injection varies from plant to plant. At present the maximum is about 40% of the coke rate achieved at Hoogovens in the Netherlands (3). Coal injection reduces the energy demand for coke making. On the other hand, the energy demand for the blast furnace increases because the coal/coke substitution ratio is on the order of 1.04 (8) to 1.25 (5) and more oxygen is required. The overall SEC will decline by 0.2 GJ/tcs (5) to 0.5 GJ/tcs (8).
3. The ore input into the blast furnace of the reference plant is 70% sinter and 30% pellets. This ratio differs greatly among countries. For instance, in the

Table 2 Specifications of the reference plant^a

Unit operation	Basic specification	Method of underfiring	Others
Coke oven (CO)	4 batteries of 100 ovens	Enriched BF ^b gas	Wet charging and water quenching
Sinter plant	2 strands of 400 m ²	Coke breeze and CO gas	Combustion air preheated in sinter cooler by heat exchange with hot sinter
Hot blast stoves	Blast heated to 1100°C and compressed to 4.0 bar	37% CO gas and 63% BF gas (enthalpy basis)	Compressor driven by condensing steam turbine Cold blast air preheated by heat exchange with exhaust to 200°C
Blast furnace	2 furnaces of 4400 m ³	Coke rate is 470 kg/tonne pig iron	Equipped with top gas pressure turbines No coal injection
Basic oxygen furnace (BOF)	2 out of 3 converters of 360 tonnes		Hot metal ratio is 75% BOF gas recovery
Continuous casting	2 twin-strand slab casters		
Reheating furnace	3 multizone walking beam furnaces	87% CO gas and 13% BF gas	Air preheat to 500°C
Hot strip mill	Fully continuous with 5 roughing and 7 finishing stands		Waste heat boilers installed
Power plant	Steam boiler, back-pressure steam turbine and condensing steam turbine	CO gas, BF gas, and BOF gas	Plants' electricity demand is exactly satisfied Medium-pressure and high-pressure steam is produced

^aBased on Reference 8.^bBF, blast furnace.

United States the sinter input is 20% of the raw material input, whereas in Luxembourg it is more than 90% (33). The 70:30 ratio seems to be a reasonable choice. Production of pellets is not included in the reference plant. Several integrated mills also have a pellet plant. The SEC for pelletizing in a modern facility is about 1.0 GJ/tonne of pellet on a primary energy basis (5). The SEC of the reference plant would increase by 0.45 GJ/tonne of rolled steel if pelletizing were to be included.

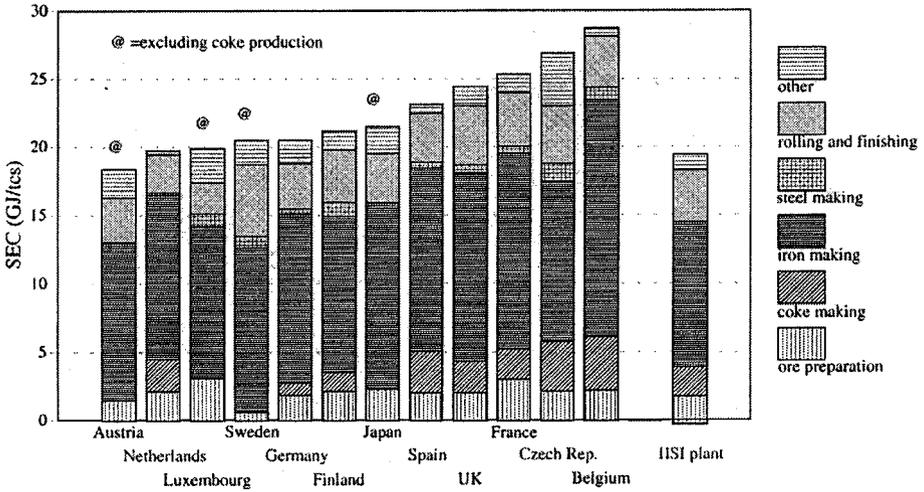


Figure 7 Countries' average specific energy consumption (SEC) (1994) for primary steel making and the SEC of the International Iron and Steel Institute reference plant. Data are derived from Reference 3. Coke making is not included in the data for Austria, Japan, Luxembourg, and Sweden (marked with @). Consequently, in these cases the SEC should be adjusted upward by about 2–3 gigajoule (GJ)/tonne of crude steel (tcs). The data for Austria are for 1993.

4. On other points there is no noteworthy difference in operational practice between contemporary mills and what is described for the IISI plant. The differences remain within the range of differences between mills.

On the basis of these observations we conclude that we are justified in using the plant described by IISI as the reference plant for our exergy analysis.

For clarity, we simplify the flow sheet of the IISI reference plant into two main points:

1. In the IISI reference plant, 58% of the crude steel is continuously cast into slabs, 29% is cast into ingots, and 11% is continuously cast into blooms. In our analysis we consider only continuous slab casting. Since the 1980s, continuous casting has become a well-accepted technique; in 1994, 72% of the world crude steel production was continuously cast (33). In modern integrated mills, ingot casting is rarely used. Continuous bloom casting resembles continuous slab casting and is therefore not treated separately.

2. The IISI reference plant considers finishing operations at a high level of detail. Several products are taken into account. Finishing operations are excluded from the analysis for two reasons. First, the energy consumption of these operations is small compared with that of the front end of the plant. Second, the

Table 3 Energy balance of the reference plant

Input				Output			
Flow	Volume	Unit	GJ/trs ^a	Flow	Volume	Unit	GJ/trs
Coal	658.33	kg	20.80	BOF ^b gas	15.71	Nm ³	0.14
Oxygen	48.51	Nm ³	0.17	CO ^c gas	43.87	Nm ³	0.85
				Coal tar	24.23	kg	0.90
				Benzole	8.56	kg	0.36
				Coke breeze	23.60	kg	0.75
Total			20.97	Total			3.00

^aGJ/trs, gigajoule per tonne of hot rolled steel.

^bBOF, basic oxygen furnace.

^cCO, coke oven.

configuration and capacities of finishing operations vary from mill to mill. The IISI configuration is not necessarily representative of other integrated mills.

These assumptions reduce specific energy consumption. From the energy balance of the reference plant in Table 3, it can be seen that the SEC is 18.0 GJ/tonne of rolled steel.

The capacity of the IISI reference plant is 8 million tonnes of crude steel per year. All figures are presented in relation to the production of 1 tonne of hot rolled steel (trs).

The composition of several flows was not given or was only partially given by IISI. We used information from literature to complete the data. An average composition of iron ore was taken from Tierney & Linehan (25). The composition of coke oven gas was taken from Szargut & Morris (34) and adjusted slightly to match the lower heating value given by IISI. The same procedure was used for coal (35) and coke (27). Finally, the composition of coal tar was taken from Spielmann (36). The lower heating values of coal and in-house-generated fuels are given in Table 4.

The exergy analysis is performed with Enerpack (37). Exergy values of flows are related to an environmental reference system (ERS). Enerpack uses the environmental temperature (in our analysis, 298.15 K) and pressure (101.325 kPa) as

Table 4 Lower heating values of energy carriers^a

Energy carrier	LHV	Energy carrier	LHV
Coal	31.6 MJ/kg	Coke oven (CO) gas	19.3 MJ/Nm ³
Coke	29.8 MJ/kg	Blast furnace (BF) gas	2.8 MJ/Nm ³
		Basic oxygen furnace (BOF) gas	9.0 MJ/Nm ³

^aLHV, lower heating value; MJ, megajoule.

Table 5 Environmental reference system values of selected chemical elements^a

Element	Reference compound	Enthalpy (kJ/mol)	Entropy (J/K/mol)	Exergy (kJ/mol)
H ₂	H ₂ O	285.69	169.34	235.20
C	CO ₂	393.52	-56.82	410.46
N ₂	N ₂	0.00	-2.31	0.69
O ₂	O ₂	0.00	-13.26	3.96
Si	SiO ₂	910.94	195.29	852.71
Ca	CaCO ₃	813.40	338.53	712.47
Fe	Fe ₂ O ₃	412.10	147.31	368.18

^aBased on Reference 37.

reference for hot and cold flows and flows with an elevated or reduced pressure. It also uses the most stable compound that occurs in the natural environment as reference for chemical elements. The chemical elements nitrogen, oxygen, and carbon and the noble gases have their reference compounds in the atmosphere. Nitrogen, oxygen, and the noble gases are themselves reference compounds. Carbon dioxide is the reference compound for the element carbon. For the remaining elements, the reference compounds are taken from the lithosphere, the hydrosphere, or a combination of both. Liquid water is taken as the reference compound for the element hydrogen. Table 5 gives an overview of the ERS values of the chemical elements that are of importance in this study. Once the exergy of the elements is determined, the exergy of all other compounds can be calculated.

PROCESS DESCRIPTION Figure 8 is a simplified flow sheet for the plant. The process starts with the preparation of the raw materials. The blast furnace requires an open structure to allow gases to ascend and liquid material to descend. For this purpose, coal is converted to coke in the coke ovens and iron ore is agglomerated in the sinter plant. An additional objective of sintering is to increase the surface reactivity of the ore. The temperatures in the coke ovens and the sinter plant are 700° and 1000°C respectively. Both coke and sinter cool to the environmental temperature before being fed to the blast furnace, together with pellets and lime. Here iron ore is reduced to pig iron, which leaves the furnace at about 1400°C. The reactions that take place in the blast furnace have already been described. During transport to the BOF plant, the pig iron cools by about 140°C. In the BOF, carbon from pig iron reacts with oxygen, injected through a lance. In the reference plant, 25% of the charge to the BOF is cold scrap, which is heated and melted in the BOF. The liquid steel with a temperature of about 1650°C is transported to the continuous caster, where slabs are cast.

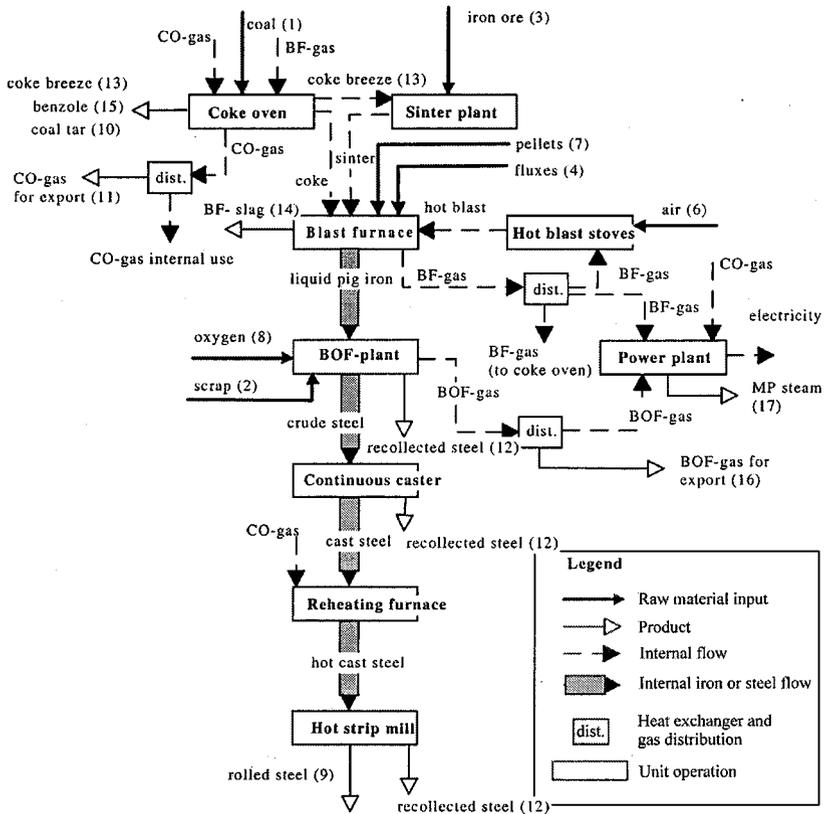


Figure 8 Flow sheet of the integrated steel mill used in the exergy analysis. Numbers (in parentheses) refer to the flows given in Table 6. Raw material and product flows are shown, as are the internal flows of product gases and iron and steel. In total, 141 flows were taken into account. CO, coke oven; BF, blast furnace; BOF, basic oxygen furnace; MP, medium pressure.

Three quarters of the cast steel cools to the environmental temperature. The remaining 25% cools to 500°C. All steel is reheated to 1200°C in the reheating furnace and then rolled in the hot strip mill. Finally, the hot rolled products cool to environmental temperature.

4.2 Results of the Exergy Analysis

Table 6 presents the results of the exergy analysis. Of the 22.6 GJ per tonne of hot rolled steel (trs) that goes into the process, mainly in the form of coal, 10.9 GJ/trs is inherited by useful products. The exergy of 1 tonne of rolled steel almost equals the minimum amount of energy needed to produce iron from

Table 6 Exergy balance of the reference integrated steel plant^a

Input	GJ/trs	Output	GJ/trs
Coal (1)	20.2	Rolled steel (9)	6.62
Scrap (2)	1.87	Coal tar (10)	0.92
Iron ore (3)	0.22	Coke oven export gas (11)	0.84
Fluxes (4)	0.20	Recollected steel (12)	0.76
LPG (5)	0.05	Coke breeze (13)	0.72
Air—various flows (6)	0.03	Blast furnace slag (14)	0.56
Pellets (7)	0.03	Benzole (15)	0.25
Oxygen (8)	0.01	Basic oxygen furnace export gas (16)	0.14
		Medium-pressure steam (17)	0.10
		<i>Total useful products</i>	<i>10.9</i>
		External losses	5.47
		Internal losses	6.15
Total	22.6	Total	22.6

^aNumbers in parentheses refer to the flows in Figure 13. External and internal energy losses are specified in Table 7. GJ/trs, gigajoule per tonne of hot rolled steel; LPG, liquid petroleum gas.

hematite, according to Reaction 4. This is due to the fact that hematite is the reference substance for iron in the ERS. The difference between exergy input and useful output, equal to 11.7 GJ/trs, is considered to be lost. External losses, i.e. losses associated with flows that are not recovered for utilization purposes, account for 5.5 GJ/trs of this loss. The remainder, 6.2 GJ/trs, is caused by exergy losses that occur within the system boundaries of the plant, the internal losses. The external and internal losses are specified in Table 7.

Radiation and convection losses are the largest source of *external losses*. This type of loss accounts for about 3.6 GJ/trs, or 30% of the total exergy loss of the steel mill. Of this, about 2.5 GJ/trs is due to exergy lost by cooling materials. The large share of convection and radiation losses will probably come as no surprise considering the large temperature differences that occur. This is illustrated in Figure 9, which shows the change in temperature and enthalpy of the solid flows. Four different material flows are distinguished.

About 1.6 GJ/trs of exergy is lost as chemical and physical exergy of waste gaseous streams. (A flow has positive physical exergy when the pressure and/or the temperature differs from the reference pressure and temperature, respectively.) In this category the exhaust of combustion reactions forms an important group. Because the chemical exergy of these waste streams is low, almost all exergy of these streams is physical exergy loss, mainly resulting from elevated temperature.

The last category of external losses is the loss of material. Coal is lost as dust that is removed from the coke oven gas. Blast furnace gas also contains dust

Table 7 Specification of external and internal exergy losses

	External exergy losses (GJ/trs) ^a				Total internal losses	Total exergy losses
	Radiation and convection losses	Chemical or physical exergy of waste streams	Material losses	Total		
Coke oven	0.28	0.47	0.24	0.99	0.87	1.86
Sinter plant	0.29	0.39		0.68	0.98	1.66
Hot blast stoves	0.25	0.11		0.36	0.41	0.77
Blast furnace	0.44	0.18	0.04	0.66	1.35	2.01
Basic oxygen furnace plant	0.12	0.06		0.18	0.34	0.52
Continuous caster	1.05			1.05	0.06	1.11
Reheating furnace	0.04	0.20		0.24	0.50	0.74
Hot strip mill	0.62			0.62	0.12	0.74
Power plant	0.20	0.21		0.41	1.51	1.92
Others	0.28			0.28	0.03	0.31
Total	3.57	1.62	0.28	5.47	6.15	11.62

^aGJ/trs, gigajoule per tonne of hot rolled steel.

(coal, iron, ore), which has to be removed before the gas can be used. The dust is considered lost. In total 0.5% by weight of the pig iron is lost with dust and blast furnace slag. Steel is lost in several operations, e.g. tapping the steel from the BOF, casting, and rolling. It is assumed that these steel losses are collected and can be reprocessed. Therefore, they are considered useful products (flow 12).

The main sources of *internal losses* are combustion reactions, other chemical reactions, heat transfer, and compression and expansion. Calculating internal losses is difficult because the underlying processes are often complex, e.g. the set of chemical reactions in the blast furnace, and because different types of internal losses are interlinked, e.g. combustion and heat transfer. Here the discussion is restricted to an estimation of the size of four different categories.

The largest amount of internal loss is related to combustion reactions. The exergy loss resulting from the irreversibility of the reaction is equal to the difference in the exergy of the reactants at input temperature and pressure and of the combustion products at the combustion temperature (27, 37). Without preheating of the reactants, this loss amounts to about 30% of the exergy content of fuel (37). In the integrated steel plant, combustion reactions take place in the coke oven, sinter plant, hot blast stoves, reheating furnace, and power plant. The exergy of all fuels, both in-house-generated gases and bought gases, is about 7 GJ/trs. The exergy loss of combustion reactions depends on the conditions in which the combustion takes place. Assuming that about 30–35% of the exergy

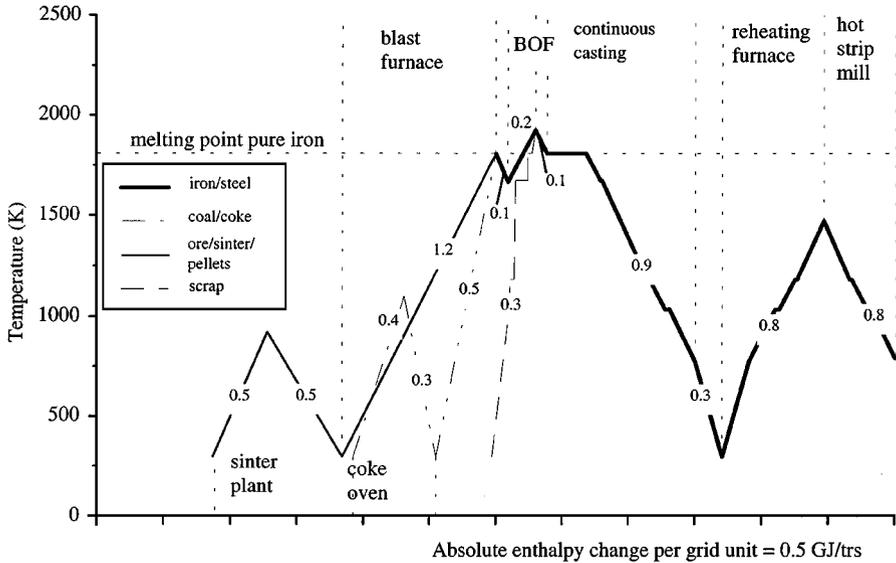


Figure 9 Enthalpy and temperature changes of the iron and steel flows in an integrated steel plant. Data on heat capacity, except for coal, are taken from Reference 26. Data for coal are based on Reference 109, in which, it should be noted, the relation between heat capacity and temperature is valid only in the temperature range between 273 and 473 K. It is assumed that above 473 K the heat capacity of coal is temperature-independent. Different slopes for the same material flow indicate that the volumes of the flows differ. Positive slopes indicate that energy has to be added; negative slopes indicate that energy is released. The numbers indicate the absolute change in enthalpy in gigajoule (GJ)/tonne of hot rolled steel (trs). BOF, basic oxygen furnace.

is lost during combustion, we can estimate the exergy losses from combustion reactions to be 2.0–2.5 GJ/trs.

A second type of internal exergy loss is caused by irreversibilities in chemical reactions other than combustion, for instance the conversion of coal to carbon monoxide. In this category we also group exergy losses that occur as a result of (a) the friction of gases with solids when they flow through a reactor (mainly in the blast furnace), (b) mixing of flows, and (c) pressure drops over reactors. The losses in this category occur mainly in the coke oven, the blast furnace, and the BOF. If we assume that all losses in these processes that are not due to combustion reactions and heat transfer are, in one way or another, related to chemical reactions other than combustion, the losses total 1.5–2.0 GJ/trs.

A third type of internal loss is caused by heat transfer, which takes place in heat exchangers and can occur simultaneously with combustion reactions. Heat

transfer can take place through surfaces or by mixing of the flows. The driving force behind heat transfer is temperature difference. The larger the temperature difference, the larger the exergy loss. However, exergy loss is smaller at high temperatures. In heat exchangers, exergy losses are usually minimized by using a countercurrent operation that ensures the smallest temperature difference. On the basis of calculations of exergy loss of the main heat transfer processes (in the power plant, in hot blast stoves, in heating ore in the blast furnace and the sinter plant, and in the coke oven), we estimate the internal losses resulting from heat transfer at 1.0–1.5 GJ/trs.

Finally, a category of other internal losses can be distinguished that contains mainly losses resulting from irreversible compression and expansion. These losses occur, for example, in the compressing of the blast, in the expansion of the blast furnace top gas, and in the power plant. These losses are estimated to be about 0.5–1.0 GJ/trs.

4.3 *Conclusions*

We can conclude that exergy losses are due mainly to the application of high temperatures and the need for several cooling and reheating steps. Radiation and convection losses, physical exergy lost with gaseous streams, losses resulting from the conversion of chemical energy to gases with a high temperature, irreversibilities in heat transfer, and even irreversibilities in some undesired chemical reactions that occur only at higher temperatures all contribute to these exergy losses. Reducing the exergy loss should therefore be directed at reducing the temperature or decreasing the number of temperature changes. In the next section we investigate whether techniques under development can achieve these objectives.

5. IDENTIFICATION AND SELECTION OF LONG-TERM ENERGY-EFFICIENT TECHNIQUES

In this section we discuss the way information on long-term energy-efficient techniques was gathered and how we selected the techniques that may reduce the SEC of steel making in the long run.

5.1 *Gathering of Information*

The identification of new techniques started with a search for relevant literature, performed in two ways. First, the following literature databases were searched: Applied Science and Technology Index, Environline/Energyline, Metadex, and Compendex. These databases were searched in two steps. At the start of the research a general search was performed. Later, when more specific key words were known (e.g. names of techniques), the searches were repeated using these

keywords. The second method of literature search was scanning volumes of journals specific to the iron and steel industry to identify emerging techniques. Of the following journals, the volumes from 1988 to 1995 were scanned: *Journal of the Iron and Steel Institute of Japan*, *Stahl und Eisen*, and *Steel Times*. We expanded our database of literature by checking the references of the collected literature.

The next step in the gathering of information was contacting the developers of the techniques to obtain the most recent data.

We checked all data for accuracy and reliability by consulting experts, and by making our own calculations and judgments, or by obtaining evidence from other sources.

5.2 Selection of Energy-Efficient Techniques

In the previous section we concluded that the main exergy losses are due to the application of high temperatures and the need for several cooling and heating steps. In current steel making, high temperatures are necessary to achieve several goals, e.g. to change the structure of the ore and coal so that they can be processed in the blast furnace, to overcome kinetic and thermodynamic limits to chemical reactions in the reduction of iron oxide, and to provide steel in a liquid form so that it can be shaped.

Techniques that reduce exergy losses resulting from high-temperature applications can be divided into three groups, according to the degree to which the need for high temperatures is avoided or reduced.

1. **TECHNIQUES THAT AVOID AT LEAST ONE HEATING AND COOLING STEP** The avoidance of one heating and cooling step can be achieved by techniques that combine two or more processes. The two major groups of techniques are smelting reduction processes and near-net-shape casting techniques. Smelting reduction processes make direct use of coal and usually also of iron ore, without having to convert coal to coke and ore to sinter or pellets. Near-net-shape casting techniques reduce or eliminate the reheating demand in the shaping of products. A completely different route involves avoiding the iron ore reduction by processing recycled scrap and subsequent melting, casting, and shaping.

2. **TECHNIQUES THAT REDUCE THE TEMPERATURES REQUIRED IN DIFFERENT PROCESS STEPS** Reduction of iron ore below the melting point is already commercially feasible in direct reduction processes. Coke making at lower temperatures is a topic of research. Casting and shaping without melting can be accomplished by powder metallurgy, a process that is already used commercially for the production of some speciality products.

3. TECHNOLOGIES THAT RECOVER AND APPLY HEAT AT HIGH TEMPERATURES
Technologies that recover and apply heat at high temperatures do not alter the need for high temperatures. In an integrated steel mill, waste heat recovery from clean gaseous flows like combustion gases is normal practice. Recovery of the heat from gaseous flows that are contaminated with, for example, organic compounds and small solid particles runs into technical problems, such as the fouling of heat exchangers. Recovery of the sensible heat from solid flows is not an important point of research interest; therefore information on this issue is not available.

6. CHARACTERIZATION OF LONG-TERM ENERGY-EFFICIENT TECHNIQUES

In this section we characterize the selected techniques. The focus is first on techniques that avoid at least one heating and cooling step. Smelting reduction processes are dealt with in Section 6.1, and near-net-shape casting techniques are dealt with in Section 6.2. Both sections start with a general description including the formulation of a general basis for comparison, i.e. the way the SEC and the costs are determined, and a description of the main production parameters. Then, separate techniques are described. Both sections conclude with a comparison of the techniques.

With regard to the techniques described in Sections 6.1 and 6.2, we assess the degree of technical change that is required to implement the technique compared with the current technique. We distinguish three categories of required technical change. First, techniques that require an *evolutionary change* imply a continuation of the trend in technological development. No changes in the way the energy service is performed are expected, and the effects on the following aspects are small or negligible: performance, process parameters, quality and nature of the products, the purchasing and supply industry, and the plant layout. Second, a *major change* is required when at least part of the energy service is performed according to a new principle, the performance of the process increases more than one can expect by trend extrapolation, and there are considerable effects on the other aspects. Finally, a *radical change* is required when a new energy service arises or all the aspects change to a large extent. (For a more extensive description of this categorization, see Reference 10.)

In Section 6.3 the state of the art and the developments in making steel from scrap are discussed. Section 6.4 deals with steel making at lower temperatures, and Section 6.5 with waste heat recovery techniques. Finally, in Section 6.6 future process routes for steel making are sketched and the potential for the reduction of the SEC is determined. In this concluding section we evaluate to

what extent exergy losses can be reduced by the techniques described, and we suggest what needs to be done to achieve further reduction.

6.1 *Smelting Reduction Processes*

Smelting reduction (SR) processes involve reduction of iron ore without the need for coke and—in most cases—agglomerated ore. The driving forces behind the development of SR processes are the reduction of capital and operation costs and the smaller environmental impact, both of which can be achieved by eliminating coke ovens and ore agglomeration.

The principle behind SR is that iron oxide is reduced in the liquid state by carbon or carbon monoxide. Liquid state reactions are much faster than solid state reactions. Because the reduction in a blast furnace is a solid state reaction, the reduction time can be reduced.

In principle, an SR process can consist of a single reactor in which unprepared iron ore and coal react to form a product similar to steel; that is, decarburization of the iron takes place in the same reactor. In practice, SR processes consist of at least two reactors, and the product resembles pig iron, which has to be refined in a separate reactor for steel to be obtained. Figure 10 gives some schematic representations of SR processes.

In SR processes, iron ore is prerduced in the solid state in a prerduction shaft by a reducing gas generated in a smelting reduction vessel. Melting and final reduction generally take place in this smelting reduction vessel as well. In many SR processes, the reaction site is the slag floating on the bath of liquid iron. Coal reacts with oxygen or iron ore in the liquid state to form a gas that consists mainly of carbon monoxide. The gas causes the slag to foam. Foaming slag is important for improving reaction kinetics and heat transfer but should be kept under a critical value to ensure normal operation. The gas can be partially postcombusted above the slag to adjust the chemical composition. The degree of postcombustion should be controlled to ensure that the composition and the temperature of the reducing gas match the requirements of the prerduction. The heat generated by postcombustion should be returned to the bath.

Three interrelated production parameters are of importance in smelting reduction processes (38):

1. *Postcombustion degree*: the degree to which the CO formed in the smelting reduction vessel by coal gasification is converted to CO₂. A too-low postcombustion degree means that the gas that leaves the reactor at the top and is used for prerduction is too rich, and a large amount of export gas is generated, resulting in high coal consumption. A too-high postcombustion degree means that the gas is too lean for prerduction and the off-gas temperature is too high (39).

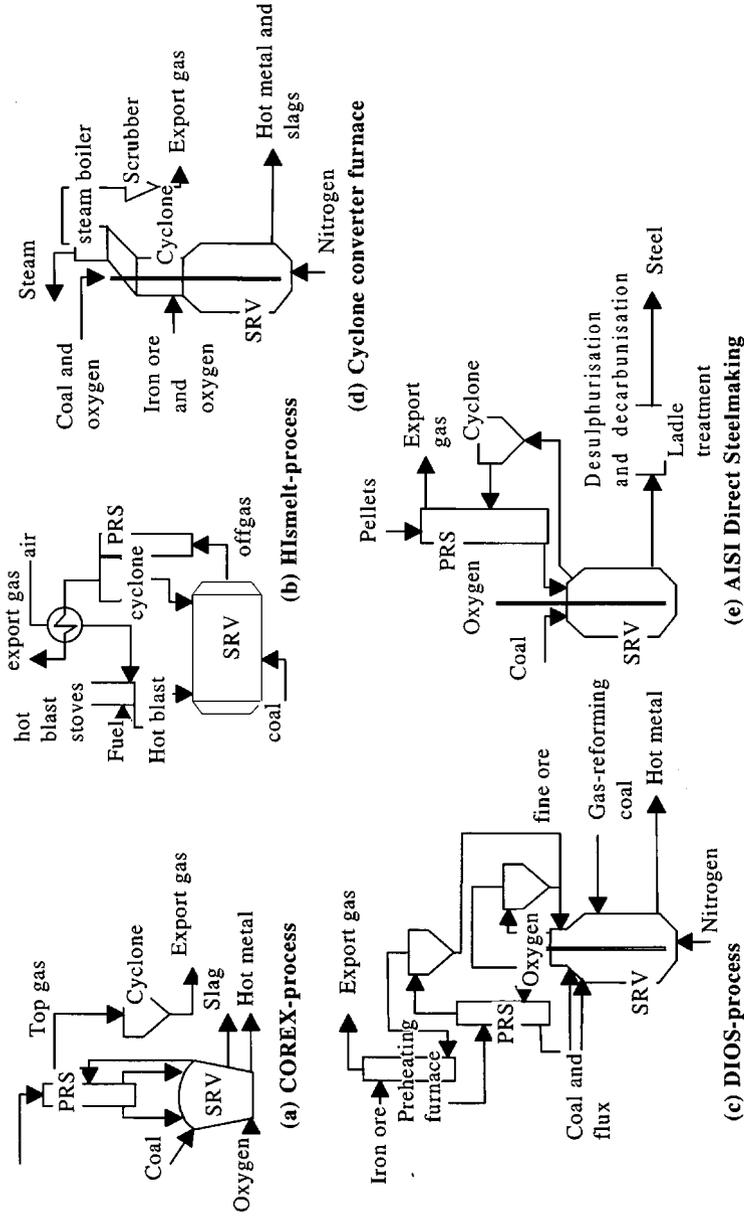


Figure 10 Schematic representations of smelting reduction processes (not on the same scale). PRS, preproduction shaft; SRV, smelting reduction vessel.

2. *Prereduction degree*: the degree to which Fe_2O_3 is reduced to Fe and FeO in the prereduction shaft (38).
3. *Heat transfer efficiency*: the ratio of the heat transferred from the hot gases to the bath of molten iron, ore, and slags and the heat generated by post-combustion (38). Too-low heat transfer efficiency results in a gas temperature too high for the constructing material of the prereduction shaft. Heat transfer efficiency is limited by the maximum attainable heat transfer from the gas to the liquid phase.

SR processes can be divided into two groups that differ considerably in the way the production parameters are controlled. Because the development of these processes also differs, they are also referred to as first- and second-generation processes (40).

First-generation processes are processes in which iron ore is prereduced to a high degree (up to 90%) before being fed to a shaftlike SR furnace. Prereduced ore and coal are present as solids in the reactor, in either a fluidized or a permeable bed. No postcombustion takes place in this furnace. The only commercial SR process, COREX, has a shaft-type furnace. Kawasaki (Japan) developed a smelting reduction process with a shaft-type furnace before it became a partner in the joint Japanese effort to develop the direct iron ore reduction process. Hoogovens (in the Netherlands) initially studied a shaft-type process (converted blast furnace), before it focused on the cyclone converter furnace.

In *second-generation processes*, the SR reactor is derived from the converter process for steel making. The final reduction takes place in a bath of molten iron and prereduced ore, with a molten slag floating on it. These processes are characterized by rapid reduction of iron ore in the molten slag layer, high heat transfer efficiency, and postcombustion of the process gases above the molten slag layer. No commercial second-generation SR process is available, although many processes have been studied or are under development. The most important of these processes are direct iron ore smelting (Japan), high-intensity smelting (Australia), American Iron and Steel Institute direct steel making (United States), converter cyclone furnace (Netherlands), Jupiter (France, Germany), and Romelt (former USSR, United States).

A group of smelting reduction processes, which can be both first and second generation, uses coal for reduction and electricity for melting, e.g. by electric arcs, plasma, or flash smelting. Examples are INRED, ELRED, and Plasmamelt. For the INRED process, it is claimed that all electricity required can be generated by using the heat of the off gas (9). However, 620 kg of coal per tonne of hot metal is required, which is about 30% more than for modern blast furnaces. The Plasmamelt process requires 275 kg of coal and coke and 1120-kWh electricity per tonne of hot metal (9). From an energy point of view,

this is competitive with modern blast furnaces only when the electricity is generated with an efficiency of almost 100%. Because these processes do not have the potential to reduce the SEC of iron making, we do not consider them further.

ASSUMPTIONS FOR COMPARISON OF SR PROCESSES Before the techniques can be compared, assumptions must be made concerning the way in which the SEC and the investment and operation costs are determined.

The *specific energy consumption (SEC)* of smelting reduction processes is presented on a primary energy basis. A breakdown into energy carriers is given. So that final energy can be converted to primary energy, several assumptions have been made. To take the sensitivity for these assumptions into account, we consider a low and a high case. The energy input of smelting reduction processes is often expressed in tonnes of coal. The lower heating value (LHV) of coal varies considerably with the composition. [Carbon, hydrogen, and sulphur contribute to an increase in heating value, while nitrogen, oxygen, ash, and water have a reducing effect (35).] Van Gool reports values varying from 20 to 38 GJ/tonne of coal (41). IISI uses 30 GJ/tonne as an average value (42). Here we use a lower limit of 29 GJ/tonne and an upper limit of 32 GJ/tonne. The composition of the export gas, thus its LHV, depends on the type of coal used. However, because the composition is often not reported, we do not consider this variation here. Another input to most smelting reduction processes is oxygen. We assume that oxygen is produced in an air liquefaction plant, with an electricity use of 280 kWh/tonne of oxygen (0.4 kWh/m³) (43). To credit for the export gas and steam produced in the SR process, we use a simple model, represented in Figure 11.

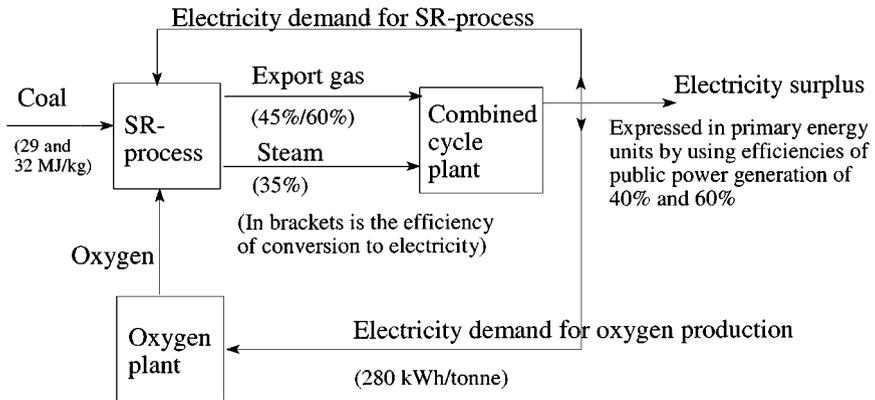


Figure 11 Model that summarizes the assumptions that were made to calculate and compare the specific energy consumption of smelting reduction (SR) processes.

We assume that electricity is generated in house from the gas produced in the SR process in a combined cycle plant. Firing low-calorific gas in a gas turbine is already a commercial technology. In Japan a combined cycle plant, fired by gas with an LHV of 3.7–4.6 MJ/Nm³, can achieve an electric efficiency of 46% (44). In September 1997 a combined cycle plant went into operation at the Hoogovens site in the Netherlands. The projected efficiency is also 46%, using gas with an LHV of about 4 GJ/Nm³ (DJC Verwey, Hoogovens Technical Engineering, personal communication). Both combined cycle plants use a gas turbine developed by Mitsubishi Heavy Industries. The LHV of export gas produced in SR processes varies from about 2 MJ/Nm³ to 7.5 MJ/Nm³. The efficiency of about 45% can also be achieved with other low-calorific gases provided that an adiabatic flame temperature of about 1100°C can be achieved (DJC Verwey, Hoogovens Technical Engineering, personal communication). To credit for the gas produced in SR processes, we assume that the chemical energy of the gas is converted to electricity in a combined cycle plant with a 45% (current) and 60% (estimate for the future) efficiency. We also assume that steam produced in the SR process is fed to a steam turbine and converted to additional electricity with an efficiency of 35% (based on 39). It is assumed that the electricity is first used for in-plant demand and oxygen generation. The surplus is expressed in primary energy units using generation efficiencies of both 40% (as a current average for electricity generation in public power plants) and 60% (future).

Because the SEC depends strongly on the method of crediting for the export gas, we perform a sensitivity analysis by varying the credit factor for the gas.

The SEC of smelting reduction processes is compared with the SEC for the production of 1 tonne of molten pig iron with the most efficient process in operation, including coke ovens, sinter and pellet plant, and blast furnace. The plant of Hoogovens is used as the reference. Figure 7 shows that Hoogovens (the only integrated steel plant in the Netherlands) is one of the most efficient plants in the world. Using the conversion factors described above, we can derive an SEC for pig iron production at Hoogovens of 16–17.5 GJ/tonne of pig iron (t_{pi}) on a primary energy basis (5). Table 8 shows a breakdown of the SEC of this reference plant into energy carriers.

A good basis for comparing the *economics* of the processes is the production cost for 1 tonne of hot metal (thm). Because hot metal is an intermediary product, exact costs are not always known. Fruehan reports a typical selling price of US\$120–140/thm of pig iron (46). Another source reports production costs on the order of US\$165/thm for a 2 million tonnes per annum (tpa) blast furnace (47). We made our own estimate of the production costs, presented in Table 9. According to our calculations, the production costs range from US\$120 to US\$160/thm. Variable costs make up 70% of total production costs, and investment costs about 30%.

Table 8 Specific energy consumption (SEC) of smelting reduction processes and of a reference process expressed in gigajoules of primary energy per tonne of hot metal^a

Process	Energy carrier	Input	Low case (GJ/thm) ^b	High case (GJ/thm)
Reference process ^c	Coal	0.59 t	17.1	18.9
	Other fuel		2.1	2.1
	Oxygen			
	Electricity demand	69 kWh	0.4	0.6
	Export gas	-3.7 GJ	-3.7	-4.2
	Export steam			
	SEC		15.9	17.4
COREX	Coal	0.88 t	25.5	28.2
	Other fuel			
	Oxygen	0.71 t	1.2	2.8
	Electricity demand	60 kWh	0.4	0.5
	Export gas	-11.5 GJ	-11.5	-12.9
	Export steam			
	SEC		15.6	17.5
Hismelt	Coal		19.3	19.3
	Other fuel		0.7	0.7
	Oxygen			
	Electricity demand			
	Export gas	-3.1 GJ	-3.1	-3.5
	Export steam	-2.7 GJ	-1.6	-2.4
	SEC		15.3	14.2
DIOS	Coal	0.81 t	23.6	26
	Other fuel			
	Oxygen	0.66 t	1.1	1.7
	Electricity demand			
	Export gas	-7.8 GJ	-7.8	-8.8
	Export steam			
	SEC		16.9	18.9
CCF	Coal	0.64 t	18.6	20.5
	Other fuel			
	Oxygen	0.67 t	1.1	1.7
	Electricity demand			
	Export gas	-3 GJ	-3.0	-3.4
	Export steam	-5.78 GJ	-3.4	-5.1
	SEC		13.3	13.7
AISI	Coal	0.7 t	20.3	22.4
	Other fuel	1.02 GJ	1.0	1.0
	Oxygen	0.56 t	0.9	1.4
	Electricity demand	62 kWh	0.4	0.6
	Export gas	-7.4 GJ	-4.4	-3.3
	Export steam			
	SEC		15.2	17.1

Table 8 (Continued)

Process	Energy carrier	Input	Low case (GJ/thm) ^b	High case (GJ/thm)
Romelt	Coal	1.35 t	39.2	43.2
	Other fuel			
	Oxygen	1.35 t	2.3	3.4
	Electricity demand			
	Export gas		pro memory	pro memory
	Export steam SEC			
Jupiter	Coal	0.57 t	16.5	18.2
	Other fuel			
	Oxygen	0.57 t	1.0	1.4
	Electricity demand			
	Export gas	-2.6 GJ	-2.6	-2.9
	Export steam SEC		14.9	16.7

^aIt is assumed that export gas and steam are converted in a combined cycle to electricity that is used for oxygen production and other electricity demand. The surplus is expressed in primary energy carriers using the efficiency of public electricity generation. Low and high cases are distinguished. Low case: electricity generation efficiency in the combined cycle plant and of the public grid of 60% and lower heating value (LHV) of coal of 29 gigajoule (GJ)/tonne. High case: electricity generation efficiency of 45% in the combined cycle and 40% of the public grid and LHV of coal of 32 GJ/tonne. Figure 11 illustrates these assumptions.

^bGJ/thm, gigajoule per tonne of hot metal.

^cThe reference process consists of coke ovens, sinter plant, pellet plant, blast furnace, and hot blast stoves. Based on Reference 5.

Only direct investment costs are considered. Annual investment costs are determined by allowing for depreciation of the investment on an annuity basis over 15 years, using real interest rates of 5% and 10%. Variable costs consist of costs for raw material, labor, energy, and maintenance. Credits are considered for the production of gas and steam.

All investment costs of SR processes are based on estimates found in literature and expressed in US dollars per tonne of hot metal. The variable costs of SR processes are determined in the same way as for the reference process. Regarding raw material costs, there are two major differences between the reference process and SR processes. First, steam coal is used instead of coking coal. Steam coal is on average US\$5/tonne cheaper than coking coal (48). The costs of steam coal are taken at US\$40–\$45/tonne. Second, oxygen is usually required instead of air. For a 1500 tonnes per day (tpd) oxygen plant, a typical capacity for SR processes, investment costs are estimated to range from US\$30 million (43) to US\$55 million (49). Reported production costs are US\$0.015/Nm³ (50) and US\$0.04/Nm³ (51). In this study the first value is used as the lower and the second as the upper limit of the costs.

Table 9 Calculation of the costs of the production of 1 tonne of hot metal (thm) in the reference plant

Investment costs	Basic data		Reference plant			
			Input (Unit/thm)	Annual costs (US\$/thm)		
	Unit/thm	Unit costs ^a (Ref.)			5% ^b	10% ^b
Blast furnace	US\$			195 (52)	18.8	25.6
Coke oven	US\$			145 (52)	14.0	19.1
Sinter plant	US\$			45 (52)	4.3	5.9
Total	US\$			385 (52)	37.1	50.6
Variable costs		Low	High		Low	High
Raw materials						
Coking coal	tonne	45 (48)	50 (48)	0.59 (68)	26.6	29.5
Steam coal	tonne	40 (48)	45 (48)			
Iron ore	tonne	25 (50)	30 (106)	1.27 (68)	31.8	38.1
Fluxes	tonne	50 (50)	60 (50)	0.1 (68)	5.0	6.0
Oxygen	Nm ³	0.015 (50)	0.04 (51)			
Labor	Manhours	10 (51)	30 (50)	0.4 (51)	4.0	12.0
Energy						
Natural gas	Nm ³	0.1 (107)	0.15 (107)	107 (68)	10.7	16.1
Electricity	kWh	0.035 (50)	0.05 (107)	70 (68)	2.5	3.5
Maintenance	% of investment	3 (51)	4 (51)		11.6	15.4
Export gas credit	GJ	-2 (51, 62)	-3 (51, 62)	3.7 (68)	-7.4	-11.1
Total					84.6	109.5
Total costs					121.7	160.1

^aUS dollars.^bInvestments are depreciated over 15 years using the annuity method with real interest rates of 5% and 10%.

The resulting production costs estimates are presented in Figure 12. The values are based on design values for raw material and energy consumption and developers' estimates of investment costs. We evaluate the effect of higher investment costs on the production costs of hot metal. Furthermore, our determination of the production costs ignores most indirect investment and manufacturing costs. Therefore, these production costs should be treated as rough estimates and used for comparison only.

CHARACTERIZATION OF SMELTING REDUCTION PROCESSES A characterization of smelting reduction processes is presented in the following SR processes: COREX, HIsmelt, DIOS, CCF, and AISI direct steel making. The Romelt and the Jupiter processes are also discussed briefly. Each characterization starts with a short history of the development, followed by a description of the

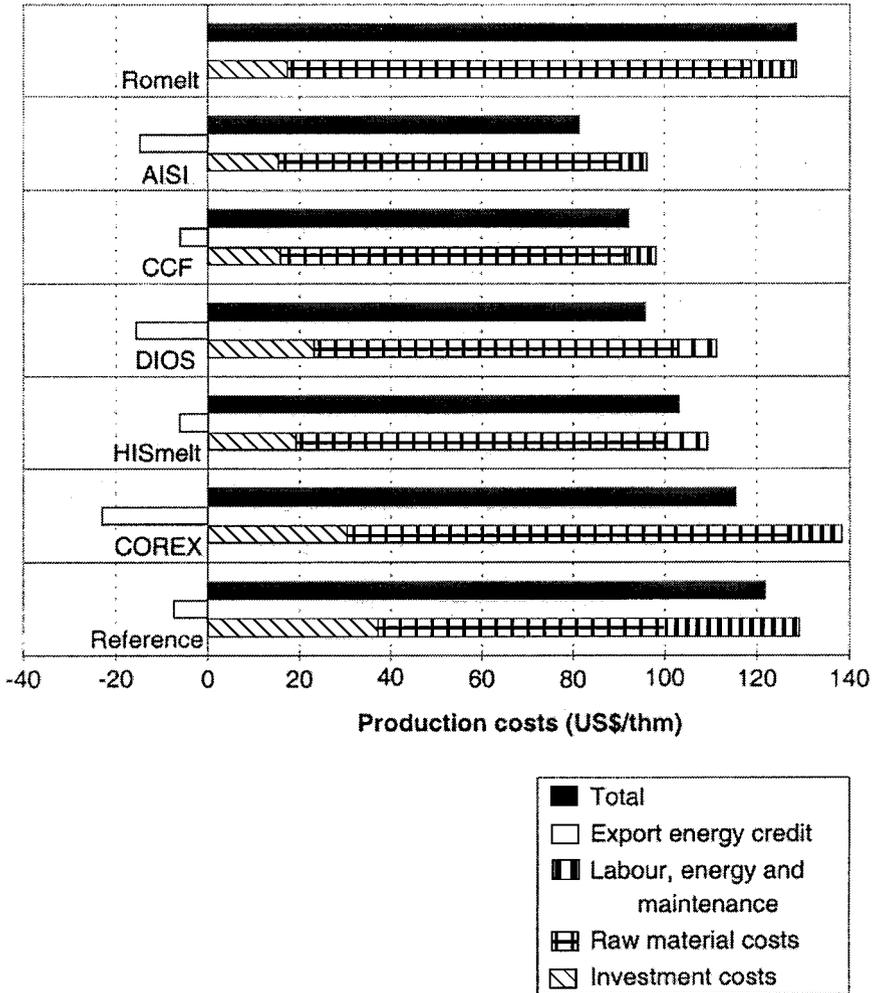


Figure 12 Production costs of hot metal [in US dollars per tonne of hot metal (thm)] of smelting reduction processes and a reference process consisting of coke ovens, sinter plant, and blast furnace. Basic assumptions are given in Table 9. The production costs represent the low case.

technique and assessment of the main process parameters. Next the SEC is determined, and finally an estimate is given of the production costs of 1 tonne of hot metal.

COREX (Germany/Austria) Development of the COREX process dates back to the 1980s and was performed by Voest-Alpine (Germany/Austria). A 300,000-ktpa commercial plant has been in operation in South Africa since 1989. This plant is named the C-1000 plant, referring to the daily capacity. Since 1995 a COREX plant has been in operation in the Republic of Korea, with twice the capacity of the South African plant (C-2000) (52). Several more orders have been placed for C-2000 plants in India, South Korea, and South Africa (52, 53).

Figure 10a gives a simplified flow sheet of the COREX process. In the dome of the smelting reduction vessel, coal is gasified in a fluidized bed, maintained by an upstream of oxygen, and injected halfway down the reactor. The coal gas reduces iron ore by 90–95% in the prereduction shaft. Prereduced iron is introduced into the fluidized bed, further reduced, and melted. The COREX process is characterized by a high prereduction degree and no postcombustion. The gas that leaves the prereduction shaft has therefore a relatively high LHV (7.5 MJ/Nm³).

Coal use in the C-1000 plant is about 900 kg/thm (54). The oxygen requirement is 540 Nm³/thm. The production of export gas is around 11.5 GJ/thm (54). A small amount of electricity is required, 60 kWh/tonne. On the basis of these data, the SEC of the C-1000 plant can be calculated to be 15.5–17.5 GJ/thm. (All calculations of the SECs of smelting reduction processes are summarized in Table 8.) First operational results of the C-2000 plant in South Korea indicate a similar SEC (55). Besides generating electricity, several other ways of using the large volumes of relative high calorific gas have been investigated, e.g. production of direct reduced iron and synthesis gas generation (52). [Two COREX/MIDREX combination plants are under construction, in South Korea (Hanbo Steel site at Asian Bay) and in South Africa (Saldanha Steel) (52, 53, 56).]

Direct investment costs of a COREX plant with an annual capacity of 1.5 million thm are US\$250/thm (52). COREX reports the production costs per tonne of hot metal to be 20% lower than that of the blast furnace process. Our calculations, presented in Figure 12, show that the investment costs are indeed about 20% lower but that the variable costs are more or less equal. As a result, production costs are estimated to be US\$115–\$150/thm, about 5% lower than for pig iron from the blast furnace.

Hismelt (Australia/Germany) The development of the high-intensity smelting reduction (Hismelt) process originates from cooperation between CRA

(Australia) and Klöckner Werke (Germany) in the early 1980s (56). A small-scale pilot plant (10–12 ktpa) was built in Germany in 1984 and operated until mid-1980 (57). In 1989 CRA formed a joint venture with MIDREX Corporation (United States) to develop the HIs melt process (58). Since 1994 a 100-ktpa pilot plant has been in operation in Kwinana in Australia (40, 57). HIs melt plans to have the first commercial plant in operation in 1999 (58); the capacity goal is 500–1000 ktpa (57). CRA's interest in the HIs melt process is related to the availability of noncoking coals in Australia and the desire to use local low-grade ores. Furthermore, the process should be able to operate in an environment where infrastructure is lacking (K Meijer, Hoogovens Corporate Research Laboratory, personal communication). This is the reason that the process is designed around the use of hot blast instead of oxygen and that the export gas production is kept to a minimum.

Figure 10*b* is a simplified flow sheet of the HIs melt process. The HIs melt process uses a horizontal cylindrical smelting reduction vessel. Noncoking coal is injected from the bottom, while prerduced ore and hot blast are injected from the top. Carbon dissolves in the bath and reacts with oxygen from iron ore. The CO formed in this way reacts above the bath with oxygen from the blast air. The generated heat is returned to the bath via a fountain of droplets of molten iron, giving a high heat transfer efficiency. The prerduction shaft is of the fluidized bed type, facilitating the processing of fine ores. The hot gases leaving the gas-cleaning cyclone are used to preheat the incoming blast air (57). The advantage of using a hot blast instead of oxygen is that the costs of oxygen production are avoided. Furthermore, the nitrogen in the air is believed to promote heat transfer and to control postcombustion temperature. A disadvantage, however, is that the size of the equipment has to be adjusted to handle the large gas volumes. The gas cleaning equipment can be especially expensive (51). Smaller gas volumes can be achieved by using air enriched with oxygen up to a maximum of 30% (58).

A postcombustion degree of more than 60% and a heat transfer efficiency of 90% have been achieved in the pilot plant (57). The high postcombustion degree is achieved by using a shallow bath with a high surface area (51). The prerduction degree is in the 20–25% range (9).

A heat balance of the HIs melt process—with all values expressed in GJ/thm—is given by Chatterjee (9). The input is 19.3 GJ/thm of coke and 0.7 GJ/thm of natural gas to fire the hot blast stoves. Then 4.0 GJ/thm of steam is generated, of which 1.3 GJ/thm is used in the turbo blowers to compress the blast air. The calorific value of the export gas is about 1.5 MJ/m³. In total 3.1 GJ/thm is produced (59). On the basis of these data a SEC of 14–15 GJ/thm is determined.

The capital costs of the pilot plant in Kwinana amounted to more than US\$100 million (58), or US\$1000/thm. The target capital cost is US\$200/thm (57). The

annual variable costs are estimated to be US\$80–\$95/thm (see Figure 12). Total production costs are US\$100–\$120/thm, about 20–25% lower than for hot metal produced in a blast furnace.

DIOS process (Japan) The Japanese direct iron ore smelting (DIOS) reduction process is being developed by cooperation between the Japanese Iron and Steel Federation, the Center for Coal Utilization, and eight Japanese integrated steel manufacturers. This national program started in 1988 and aims to have a first commercial unit in 2000 (60). The process should be available around 2010–2015 at the latest, when most of the Japanese coke ovens will have completed their 40 years of service (61). A 180,000 tonnes per year pilot plant operated in the period 1993–1995. The capacity of commercial plants that is aimed for is 1–1.8 million tonnes per year (60).

The DIOS process has three fluidized bed furnaces, one placed above the other: On the top is a preheating shaft, in the middle the prereluction shaft, and at the bottom the smelting reduction vessel (60) (see Figure 10c). Ascending gas fluidizes materials in the furnaces. In the prereluction shaft, iron ore, preheated in the preheating furnace, is prereluced by means of the gas generated in the smelting reduction vessel. The bath in the smelting reduction vessel consists of three layers. The top layer is a mixture of coal and prereluced iron ore. Final reduction takes place in the middle layer, which is molten slag and coal combustion products, e.g. chars. Molten iron sinks to the bottom layer. Oxygen is blown through a lance into the first two layers, but partial side-blowing can enhance the postcombustion degree (9). The reducing gas, generated by coal gasification in the top layer, can be reformed by additional coal injection so as to increase its reduction capacity and reduce the temperature of the reducing gas (51).

The prereluction degree is 25%, of which 5% is achieved in the preheating furnace. The postcombustion degree is 40–60% (9). The heat transfer efficiency is not known.

The SEC has not been published so far, but it is said to be lower than 16.5 GJ/thm (51). However, a SEC of 17–19 GJ/thm is determined using values given by Stelco (51). Inputs are 814 kg/thm coal and 504 Nm³/thm oxygen. The production of export gas amounts to 7.8 GJ/thm (51).

Investment costs have not been published. The costs of the pilot plant were US\$70 million (60). Investment costs of a commercial green field DIOS plant, including oxygen plant, are 80–85% of a green field coke oven/sinter plant/blast furnace. The costs of the latter are US\$385/thm (62). A DIOS plant with a capacity of 1.5 million tpa requires an oxygen plant with a capacity of 2700 tpd. The investment costs of such a plant are about US\$80 million (49). The investment costs of a DIOS plant can be estimated to be US\$230–\$250/thm.

Variable costs are estimated to be US\$75–\$95/thm (see Figure 12). Total production costs are determined to be in the range of US\$95–\$125/thm, which means that they are about 20% lower than for hot metal from a blast furnace.

Cyclone converter furnace (Hoogovens, the Netherlands) The development of the cyclone converter furnace (CCF) started in 1986. From 1986 to 1992, Hoogovens worked with British Steel and Ilva (Italy) (since 1988) on the development. Initially, a shaft-type SR process was being studied. However, because ore agglomeration was still required, a shift was made to the development of a converter type of process in 1989 (63). In 1992 British Steel decided to focus on direct coal injection and stopped their involvement in the development of CCF. Since 1994 Ilva and Hoogovens have continued their research separately (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication). Ilva operated a 5 tonnes per hour (tph) pilot plant (56). Tests by Ilva appeared to be less successful, and plans for a new series of experiments were stalled because of the high financial risk (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication). Hoogovens still continues its research and development connected with the CCF.

The CCF being developed by Hoogovens consists of a melter/prereduction cyclone mounted on a converter-type vessel (see Figure 10*d*). Granular coal and oxygen are injected through a lance into the bath of molten metal. The gases produced rise and mix with the fine ore and oxygen that are injected tangentially into the cyclone. Here, the ore is not only prereduced but also molten. The molten prereduced ore trickles down to the bath, where the final reduction takes place. Postcombustion takes place not only directly above the bath but also in the cyclone, which allows for efficient heat transfer.

In 1994 a 20-tph test facility for the melting cyclone was built and successfully operated at the Hoogovens site. The converter has not yet been tested on a pilot-plant scale. In 1995 Hoogovens acquired the rights of the AISI direct iron-making technology, which is directed primarily at in-bath smelting. Hoogovens aims to achieve plant capacities of 500 and 1000 ktpa (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication). In 1997 the Dutch government decided to support the development of the demonstration plant with 60 million Dutch guilders (US\$32 million) (64). The demonstration plant with a final capacity of 700 ktpa should be on stream in 2000 (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication).

A prereduction degree of 20% on average was obtained in the pilot-plant experiments with the cyclone alone. The postcombustion degree is expected to be 25% and heat transfer efficiency 80% in the bath, but this has not been tested. A lower postcombustion degree can be maintained than in other SR processes,

because the iron ore is not only reduced but also melted in the prereluction shaft. The direct connection between the smelting reduction vessel and the prereluction cyclone permits optimum use of the heat of the postcombustion gas, since no hot de-dusting and cooling are necessary. Since postcombustion also takes place in the cyclone, the final postcombustion degree is more than 75% (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication).

Design values for the energy input are 640 kg/thm coal and 510 Nm³/thm oxygen. Energy output is 3.0 GJ/thm export gas and 5.7 GJ/thm steam. A SEC of 13–14 GJ/thm is estimated on the basis of these data (65).

The investment costs of the CCF process are estimated at US\$150–\$180/thm (65). These low-investment costs seem reasonable considering that the CCF process is simple compared to other smelting reduction processes, mainly because the prereluction shaft and the smelting reduction vessel are combined. The variable costs are estimated to be US\$70–\$90/thm. Total production costs are estimated to be US\$90–\$115/tonne, which means they are about 30% lower than for producing hot metal in a blast furnace (see Figure 12).

AISI direct steel-making process (United States) The program of the American Iron and Steel Institute (AISI) and the US Department of Energy (DOE) for developing the direct steel-making process ran from 1988 to 1994 (66). A number of universities and industries participated in the research (67). Steel manufacturers were also closely involved. The aim of the program was to prove the technical and economic feasibility of a direct steel-making process, including continuous desulphurization and decarbonization and ladle treatment. However, early in the program the focus shifted to direct iron making, particularly to in-bath smelting. Tests in a 15-tpd pilot plant showed that smelting of wustite in a high-intensity bath process is a manageable process (67). The productivity and the fuel rate, however, were far behind the goals formulated at the start of the program (66). Nevertheless, several steel companies believed that the technical problems could be overcome and considered a 350-ktpa demonstration plant. This plant has not yet been built. Instead, AISI and DOE launched a new program to determine the feasibility of converting steel plant waste to pig iron with the in-bath smelting technology to be used in steel making and foundries (56, 66). The development of the direct iron making from ore process stalled (66). However, as mentioned in the discussion of CCF, Hoogovens is considering the use of the AISI technology in connection with its cyclone reduction and melting technology.

The AISI process involves the melting of partially prereluced iron ore pellets. Figure 10e is a simplified flow sheet. Final reduction takes place in foaming slag above the molten iron bath. Coal is top-fed into the slag. Oxygen is injected

through a lance. The carbon monoxide that is formed is postcombusted above the slag layer. The aim of the AISI technology was to prereduce pellets to wustite (67), implying a prereduction degree of about 50%. To match the reducing potential of the off-gas with this prereduction degree, a postcombustion ratio of 40% is required. In the pilot-plant tests, it has been proven that this ratio is attainable (67). The heat transfer efficiency has not been reported. The AISI process is distinct from other smelting reduction processes in that pellets are used rather than ore and that the export of gas is minimized—at least in intention. The background to the use of pellets is that in the United States a modernization of the pellet plant capacity had just been completed. Minimizing export gas production should increase hot metal production per unit of working volume per day.

Design values for the coal input of 700 kg/thm and oxygen use of 430 Nm³/thm have been reported (51, 67). The production of export gas amounts to 7.4 GJ/thm. We can estimate the SEC to be 15–17 GJ/thm on the basis of these data. In the SEC the energy demand for pelletizing is included; this amounts to about 1.5 GJ/thm (68).

Faure reports investment costs of US\$160/thm (6). Variable costs are estimated to be US\$65–\$85/thm (see Figure 12). Total production costs are therefore about US\$80–\$105/thm, or 35% lower than for hot metal produced in the blast furnace.

Other developments Besides the above-mentioned techniques, R&D has been done on several other SR processes. Because little information about these processes is available, the discussion here is limited.

Romelt process (former USSR, United States) The Romelt process is a single-stage SR process. Coal and ore are fed to a horizontal large-volume vessel. Oxygen is injected through side tuyeres at two levels, enabling a highly agitated bath and a high postcombustion degree of 70%. Reduction takes place in a foamy slag layer. This process is based on the Vanyukov process for copper smelting (9, 56).

An 18-tph pilot plant operated at the Novolipeski Metallurgical Kombinat (NLMK) in Lipetsk from 1985 to 1988. A 1000-ktpa demonstration plant was approved but never realized (56). Because of the collapse of the USSR, Soviet R&D stopped. In 1995 the engineering and constructing company ICF Kaiser International (United States) and Nippon Steel Corp (J) were granted licences to market and commercialize the process (69). ICF Kaiser believes that the Romelt process is ready for commercialization, especially for processing steel plant wastes (69). The energy input is relatively high (50). Pilot-plant results showed a minimum coal input of 1350 kg/thm and an oxygen demand of 1900 Nm³/thm (78% O₂ in blast). Although no figure for export gas production is given, the SEC of the Romelt process is probably considerably higher than the SEC of other

SR processes and the SEC of the blast furnace route. An economic analysis of a 400-ktpa Romelt plant yielded investment costs of US\$180/thm (50). Variable costs are in the range of US\$110–\$155/thm, or 30–40% higher than that for pig iron produced in a blast furnace. Total production costs are estimated to be US\$130–\$180/thm, or 5–10% higher than that for pig iron produced in a blast furnace. When steel plant waste oxides are processed, as ICF Kaiser aims to do, the production costs are reduced to US\$85–\$130/thm, if no costs are assumed for the waste oxides.

Jupiter (France, Germany) The Jupiter process was developed by IRSID, the research center of Usinor-Sacilor (France), in cooperation with Thyssen Stahl and Lurgi (both Germany) (70). However, R&D seems to have stopped (66) (K Meijer, Hoogovens Corporate Research Laboratorium, personal communication). The aim was to develop a process to supply virgin metal to EAF plants (70). A coal input of 570 kg/thm and an oxygen demand of 435 Nm³/thm were reported. Export gas production was 2.6 GJ/thm (62). The SEC can be estimated to be in the range of 15–17 GJ/thm. No cost figures are available for the Jupiter process.

Other processes Numerous other smelting reduction processes—not considered here—have been or are being developed. For instance, besides being involved in the DIOS program, Kawasaki Steel in Japan is developing the Kawasaki XR process. The Kawasaki XR process has a shaft-type furnace and is now being developed to process BOF dust (9, 51). Another example is the Chinese effort to develop a smelting reduction process without postcombustion and foaming slag (71). Iron ore concentrate is prerduced and deposited along with fine carbon in a separate reactor. In the next step, the deposited carbon is combusted quickly, resulting in flash smelting of the ore. The melted ore falls on a hot coal surface. This process is still at the stage of applied research (71).

CONCLUSIONS CONCERNING SMELTING REDUCTION Now that several SR processes have been characterized, a founded estimate can be made for the potential of energy-efficiency improvement when these processes are implemented and a comparison can be made of the production costs. We also evaluate the chance of successful commercialization of the SR processes that are still under development and estimate how long it will be until the first commercial plant is in operation.

Table 8 presents an overview of the SECs of the SR processes and the reference process. The SEC of smelting reduction processes—except the Romelt process—varies from 13 to 19 GJ/thm. The CCF process seems to be the most energy efficient; the SEC is about 20% lower than the SEC of the reference process. The HIs melt and Jupiter processes have SECs that are 5–10% lower

than those of the reference, whereas the AISI, COREX, and DIOS processes have SECs equal to or even higher than the reference.

The SEC depends strongly on the production and utilization of the export gas. Figure 13 shows how the SEC varies when the export gas credit factor is varied from 0.4 to 0.6. (The export gas credit factor is the efficiency at which the gas is converted to electricity in a combined cycle. Therefore, credit factors higher than 0.6 are not considered. This is not to say that higher credits cannot be achieved, e.g. by substitution of fuel oil in heating or as reducing gas in DRI production. However, in these cases another way of crediting should be considered. This may provide new insights, but it is beyond the illustrative scope of this sensitivity analysis.) This is done for two efficiencies of public power generation: 60% is an estimate of the maximum achievable efficiency for a gas-fired power plant, and 50% is an estimate of the maximum achievable efficiency for a coal-fired power plant. The figure shows that the SR processes can be divided into two groups according to the sensitivity of the SEC to the export gas credit factor. Processes in the first group—COREX, DIOS, and AISI—have relatively large export gas production (more than 7 GJ/thm) because of a low degree of postcombustion. Therefore, the SEC depends on the export gas credit factor. The use of the export gas should be carefully considered. For instance, Figure 13 shows that a credit factor of 0.5 is required for the AISI process to be more efficient than the reference. This is on condition that electricity can be generated in central power stations with an efficiency of 60%. When we use a 50% efficiency of power generation, AISI is already more efficient with an export credit gas factor of 0.45. Processes in the second group—Hismelt, Jupiter, and CCF—are in all cases more efficient than the reference.

The conclusion that emerges is that SR processes are not necessarily more efficient than conventional iron making. The energy requirement for coke making and, in most cases, ore agglomeration is avoided. Energy consumption of the iron ore reduction itself increases, as a result of the higher coal consumption and the need for pure oxygen. The energy consumption can be minimized by selecting optimum values for the process parameters. Careful attention should be paid to the utilization of the export gas, both in the reactor and outside. The maximum energy-efficiency improvement appears to be about 20% compared to the current best-practice iron-making process. However, all data on energy requirement are still based on design values or on pilot-plant results. But SR technology is still in an early stage of development, and further work on these technologies might well lead to even more energy-efficient designs.

Figure 12 gives an overview of the production costs of 1 tonne of hot metal of the SR processes compared with the reference process. All second-generation processes—except the energy-intensive Romelt process—have production costs that are 20–35% lower than those of the reference. This reduction

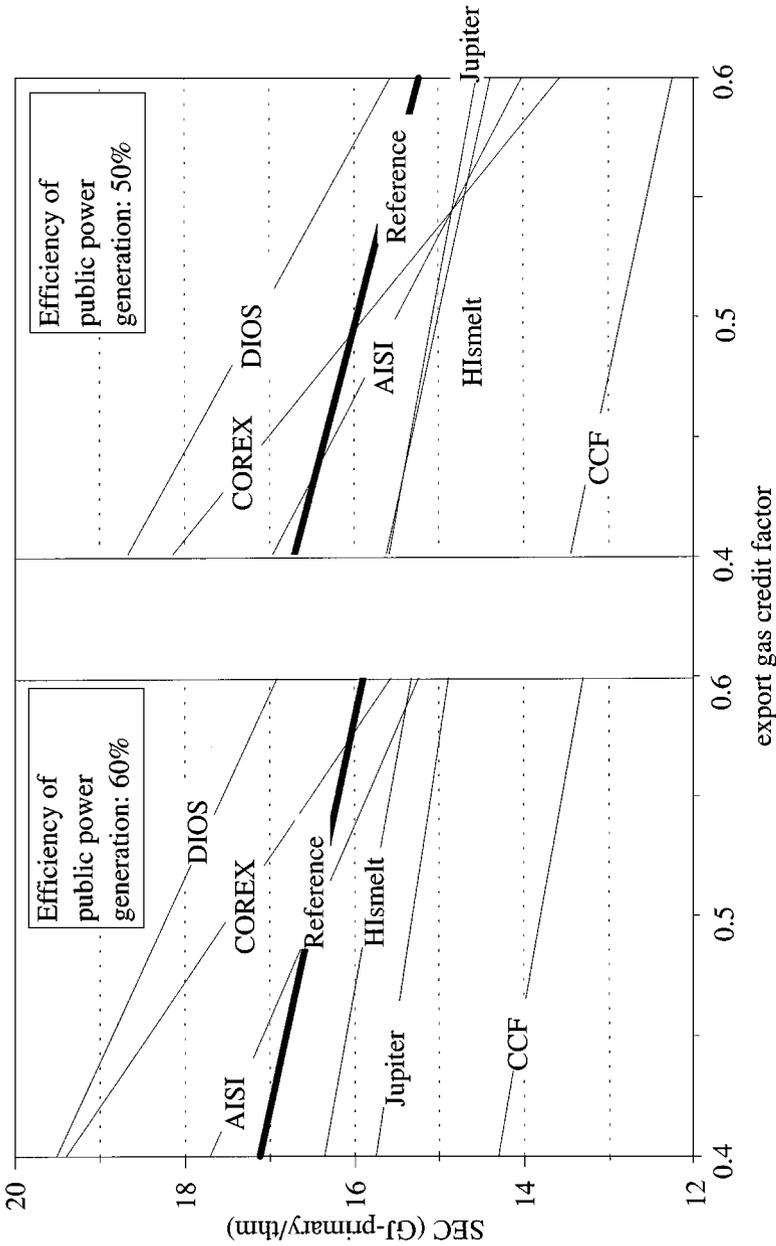


Figure 13 Dependency of the specific energy consumption (SEC) of smelting reduction processes on the credit that is given to the export gas. The export gas credit factor is the efficiency of the export gas to electricity in a combined cycle plant. For comparison, gas with a lower heating value of about 4 megajoule (MJ)/Nm³ can be converted to electricity with an efficiency of 46%. The generated electricity is used for the plant's demand. The surplus is credited for by expressing it in primary energy units, assuming an efficiency for public power generation of both 60% and 50%. GJ, gigajoule.

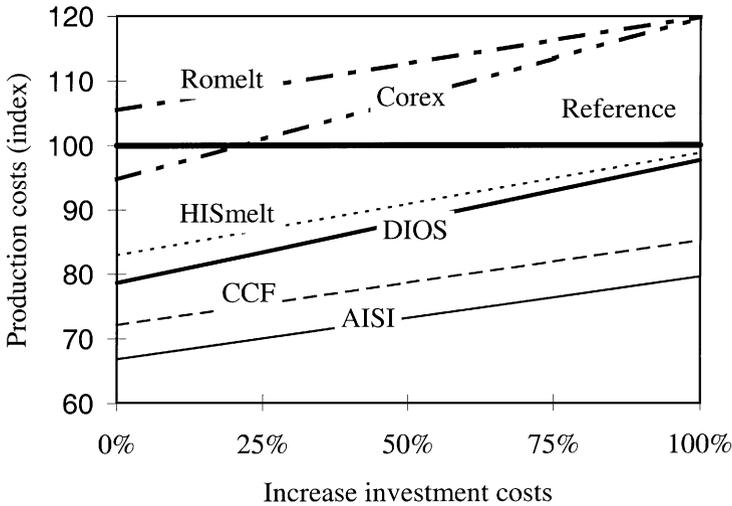


Figure 14 Change of the production costs when the investment costs, as assumed in the descriptions of the processes, are increased. The investment costs of the reference are kept constant. The production costs are calculated using the low prices.

is achieved predominantly by a 35–55% reduction in investment costs. The variable costs are 5–25% lower. The expected cost reduction by using steam coal instead of the more expensive metallurgical coal is partially offset by the larger coal demand. To get an idea of the sensitivity of the production costs of hot metal to changes in the investment costs, we increased the investment costs of the SR processes by up to 100%. We kept the investment costs of the reference process at the same level. The results are shown in Figure 14. The production costs of hot metal in the COREX process become higher than those of the reference process, if the investment increases by 25%. However, since the COREX process is already commercial, the investment costs are based on actual figures. An increase of 25% is not to be expected. The production costs of all other SR processes (except the Romelt process) remain below those of the reference up to a 100% increase of the investment costs. The sensitivity to changes in variable costs is taken into account by calculating with low and high costs (see Table 9). The high cost figures resulted in a 0.5% increase in the production costs for the DIOS process and a 7% increase for the HISmelt process.

It can be safely concluded that the production costs of hot metal in a second-generation SR process will be lower than in conventional iron making.

What chance is there that a SR process will be commercialized, and when can we expect the first commercial plant? To answer this question we assess

the technical change required to bring the technology to commercialization and give a résumé of the stage of development.

As far as the technical change is concerned, a distinction can be made between first- and second-generation SR processes. Second-generation processes require a new type of reactor, whereas for first-generation processes the blast furnace can in principle be converted to serve as a smelting reduction vessel. Industry has less technical experience with the converter-type reactor than with the shaft-type reactor. On the other hand, the technology for smelting reduction differs considerably from that needed in blast furnaces, although the principle of iron ore reduction is more or less the same. The front end of the plant, i.e. all processes up to the blast furnace, has to be replaced completely. The processes after the blast furnace can remain essentially the same.

Considering the large process adaptations, we can say that changing from a blast furnace to a smelting reduction process is a major technical change and that first-generation smelting reduction technology is a smaller technical change than second-generation technology. In Figure 15 this information is combined with the stage of development.

There are large differences in the stages of development. The only first-generation process, COREX, is already commercial. Of the six second-generation processes, only three have ongoing research into reduction of iron ore, namely DIOS, HIs melt, and CCF. HIs melt and DIOS seem to be in the most advanced

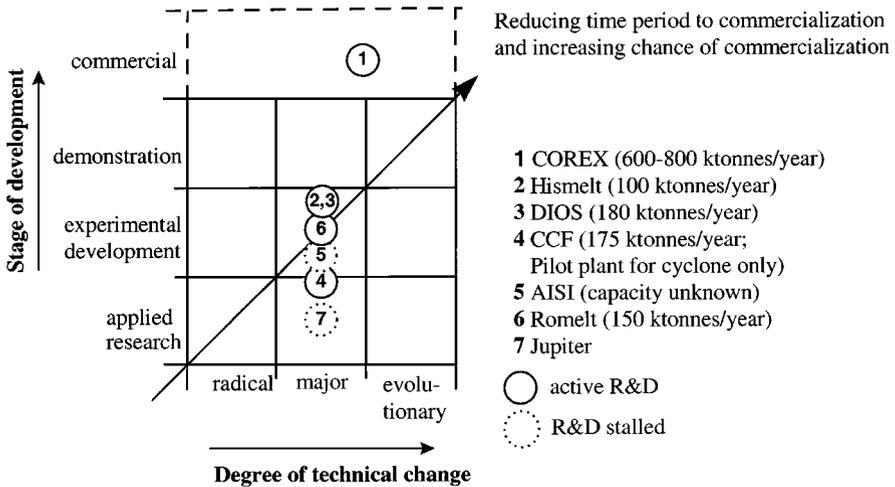


Figure 15 Comparison of smelting reduction processes with respect to the degree of technical change compared with the current technology and the stage of development.

stage of development. Both processes have been fully tested on a pilot-plant scale. The developers expect the first commercial unit to be in operation in 1999 (Hismelt) and 2000 (DIOS). The CCF has been only partially tested on a pilot-plant scale, namely only the smelting cyclone. A demonstration plant is planned to be in operation in 2000. The Romelt and AISI processes are now being developed to process steel plant wastes. However, there may still be a future for the AISI technology in ore reduction, as Hoogovens has acquired the rights and intends to combine it with the CCF. The Jupiter process has never been tested on a pilot-plant scale.

From Figure 15 it can be concluded that at least three “second-generation” smelting reduction processes are on the verge of being demonstrated. It can reasonably be expected that one or more of these processes will result in a commercial plant within the next decade.

6.2 *Near-Net-Shape Casting*

The second group of techniques that avoid at least one heating and cooling step is concerned with casting and shaping of steel. Traditionally, steel is cast into ingots of different shapes and weights ranging from several tonnes to about 300 tonnes. Nowadays, more than 60% of the crude steel is cast directly into blooms (square blocks with an outline of 0.15–1 m) and billets (small bars with an outline of less than 0.15 m) or slabs 0.15–0.2 m thick using a continuous caster (33). Blooms and billets are further processed in hot rolling mills to long products to change the shape into, for example, beams, profiles, and rails. Slabs are converted to flat products in a hot strip mill or hot plate mill to reduce the thickness to 1–10 mm for strips and 10–25 mm for plate. The thickness of flat products may then be further reduced to about 0.1–3 mm in a cold rolling mill. The casting and shaping process is characterized by its discontinuity, requiring intermediate storage and putting high demands on logistics.

Near-net-shape casting processes use techniques that can attain the final shape with fewer operations, or even in one step. The main advantages of near-net-shape casting are (a) reduction in investment and operation costs; (b) reduction in processing time between casting and final product; (c) reduction in intermediate heating and cooling and storage; and (d) improved (surface) properties resulting from a finer, more homogeneous microstructure.

The state of the art in near-net-shape casting is thin slab casting: Slabs are cast with a thickness of 40–90 mm. Thin slab casting has been applied successfully since 1985 in connection with EAF steel plants. With a combination of an EAF and thin slab caster, flat products can be produced at costs that are competitive with the costs of flat products made in an integrated steel plant. This opened the market for flat products for EAF steel, a market that had been restricted to

integrated steel producers. For a few years now, the technique has been applied in integrated steel plants as well. It is estimated that 10% of the world hot strip is produced using a thin slab caster (72). The direct casting of beams is also a commercial technique (73). This section focuses on the near-net-shape casting of flat products.

Four categories of near-net-shape casting techniques can be distinguished for flat products:

1. Thin slab casting: thickness range 40–80 mm
2. Thin slab casting with liquid core reduction: thickness range 10–25 mm
3. Strip casting: thickness range 1–10 mm
4. Spray casting: thickness range 5–20 mm

The first two techniques resemble the continuous caster and still require a reheating furnace, albeit with a smaller heating capacity. The third technique makes the hot strip mill redundant and is therefore interesting from the point of view of energy conservation. Spray casting produces semifinished products of different geometry by spraying and rapid solidification of small metal particles onto a substrate surface (74).

SPECIFIC ENERGY CONSUMPTION The integrated steel mill analyzed in Section 4 is used here as the reference process. In this process, casting requires only a small amount of heat for preheating of the ladle and electricity for, for example, drive power and crane handling. Shaping requires the most energy, particularly the reheating furnace (1.82 GJ/trs). The electricity required for driving the machines is 0.37 GJ/trs. Steam (0.15 GJ/trs) is generated in the cooling section of the hot strip mill. Presented in primary energy units, this is 2.4–2.7 GJ/trs, or about 15% of the total primary energy demand of the integrated mill. We assume that electricity is produced in a central power plant with efficiencies of 40% and 60%, respectively. We credit the generated steam with a factor 0.35, representing the ratio between exergy and enthalpy of the steam.

ECONOMICS Direct investment costs of conventional casting and shaping of hot strip can be divided as follows: 71% to the hot rolling mill and reheating furnace, 22% to the caster, and 7% to finishing (75). Absolute investment costs are not available.

Reported production costs for conversion from crude steel to hot rolled strip for an integrated mill (at the Great Lakes, United States) are US\$67/tonne of hot rolled strip (76). About 30% of these costs are for maintenance, 20% for

energy, and 15% for labor. The remainder consists of costs for replacements of rolls, for refractories, and for supplies and of credit for yield losses.

Both investment costs and operation costs of near-net-shape casting techniques are presented with reference to conventional casting and shaping per tonne of hot rolled strip.

Thin slab casting Although thin slab casting (TSC) is already a commercial technique, this technique is discussed here because experience with TSC in integrated steel mills is still small. Compact strip production of SMS Schloemann-Siemag, inline strip production of Mannesmann Demag, and continuous thin slab casting and rolling technology (Conroll) of VAI are the major TSC techniques, but others are available or are being developed (51, 74). In 1995 about 15 thin slab casters had already been installed worldwide or had been ordered, and several pilot plants were in operation (G Flemming, SMS Schloemann-Siemag, unpublished data) (74). Most of these installation are combined with an EAF. However, construction of the first facility to combine a BOF with a compact strip production plant started operation in 1994 in the United States (77). Recently, two European integrated steel manufacturers, Hoogovens (Netherlands) and Thyssen (Germany), announced that they will install thin slab casters with annual capacities of 1.2 and 1.5 million tonnes of strip, respectively (72, 78, 79).

A TSC facility resembles a continuous caster. The casting mold, which gives shape to the cast steel, is adapted to cast slabs with thicknesses of 40–50 mm (compact strip production), 30–60 mm (inline strip production), and 70–90 mm (Conroll). TSC facilities combine the caster and the rolling mill in one plant by tight control and the use of a reheating furnace that brings the steel to a uniform temperature of about 1100°–1200°C (80). The type of furnace differs depending on the configuration: SMS and Conroll apply a hearth furnace, and inline strip production uses an induction furnace. After the rolling mill, a thickness of less than 2 mm can be achieved. Industrial plants are designed with slab widths of 900–1700 mm (G Flemming, SMS Schloemann-Siemag, unpublished data). Thin slab casters can produce different types of steel, including stainless and carbon steel.

The capital costs (per tonne of product) are 30% (81) to 55% (82) of the costs for a conventional caster with hot strip mill. The largest reduction is achieved when a two-strand TSC plant is used with an annual capacity of about 1.5 million tonnes of rolled steel. At that capacity the hot rolling mill is used optimally (75). Total investment costs for the TSC plants that will be installed at Hoogovens and Thyssen are US\$200/thm and US\$300/thm, respectively. Operation costs vary from 80% to 110% of the reference, depending on capacity utilization (75, 82). TSC is already competitive at half of the capacity at which a hot strip mill operates most cost-effectively (i.e. 4 million tpa) (82).

The SEC of the inline strip production process is 196 kWh/tonne, including electricity for the induction furnace (83). The fuel requirement for reheating is eliminated in this case. In terms of primary energy, a SEC of 1.2 to 1.8 can be achieved or a reduction of 35% to 50%. For comparison, savings of 50% on the primary SEC for the compact strip production are reported (81).

TSC is a continuation of the developments in continuous casting. The quality and nature of the products are comparable to those of flat products from a continuous caster. The products can be further processed in the existing cold rolling mill and finishing operations. TSC can be integrated into existing steel plants, including integrated steel plants and plants with an EAF. On the basis of these considerations, TSC can be seen as an evolutionary change.

Thin slab casting with liquid core reduction (TSC with LCR) Slabs with a thickness of less than 25 mm can be cast by compressing the cast steel shortly after it leaves the mold, i.e. while the edges are already solid and the core is still liquid. To roll strips of 1–10 mm, a reheating furnace and rolling mill are still required. Several techniques are under development. Inline strip production also uses liquid core reduction, but the thickness is reduced from only 60 mm to 45 mm after casting (82).

Thyssen Stahl (Germany) and SMS (Germany), and partially Usinor-Sacilor (France), are working together to develop the casting pressing rolling (CPR) (84). A pilot plant was built in Germany and has been in operation since 1993 (84). After use of the pressing rolls that squeeze the just-cast steel, the strip thickness is 10–15 mm (82).

The capital costs (per tonne of output) of CPR plants are said to range from 100% to 120% of the reference (82). Kruger reports capital costs of about US\$260/trs for a CPR plant with a capacity of 450 ktpa (84). At this capacity a CPR plant is competitive with a conventional caster and rolling mill with a capacity of 4 million tpa (82). Operation and maintenance costs are comparable to the reference (82).

No data on energy demand are available. As the thin slabs have hardly any hot core, the heat demand for homogenizing the temperature is expected to be more or less the same as that for thin slab casting without liquid core reduction. The temperature increase is about 20% of that needed in a conventional reheating furnace. On the other hand, since less rolling is required, savings on electricity are expected to be higher than with TSC without LCR. We estimate the heat demand to be 20% of the reference and the electricity demand to be 80%, resulting in a SEC of 0.9–1.1 GJ/trs based on primary energy units. This is a reduction of 60–65% compared to the reference.

The step from TSC to TSC with LCR is not large as far as degree of technical change is concerned. Therefore, we use the same categorization as for TSC: evolutionary change.

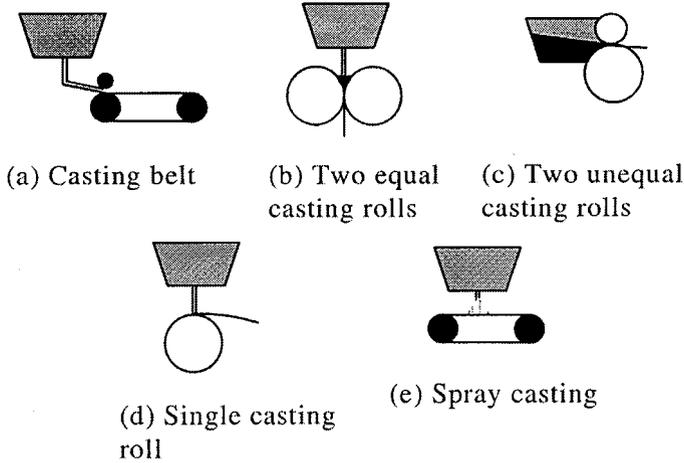


Figure 16 (a–d) Four possible configurations for strip casting and (e) one for spray casting (sketches adapted from Reference 82).

Strip casting Contrary to continuous casting (and thin slab casting), strip casting does not require a casting mold. Instead the liquid steel is cast directly on a belt or on rolls. Separate rolling is no longer required. Figure 16 gives an illustration of some possible configurations. The configuration with the two casting rolls (Figure 16b) closely resembles the strip casting machine that Bessemer patented around 1860.

Many R&D efforts have been made with regard to the development of strip casting. R&D is generally performed by a joint venture between a casting machine manufacturer and a steel manufacturer (46). Examples are Allegheny Ludlum and Voest Alpine, CSM, and Ilva. More than 30 R&D projects have been reported; most of them are still in the pilot-plant stage with ladle capacities of less than 3 tonnes (51). However, at least six strip-casting development projects are already in the demonstration stage; i.e. a ladle capacity of more than 10 tonnes can be handled (74). Commercial strip casters should have a ladle capacity of 20–25 tonnes. So far, no commercial strip casters have been developed. Problems with the geometry of the strip appear to be the main bottleneck for development, although surface quality and mechanical-technical properties of cold rolled cast strip also require serious R&D. Furthermore, production of carbon steel, the most important type of steel, is difficult. Differences in steel microstructure and resistance to oxidation are characteristics that make stainless steel favorable to carbon steel (51). However, some companies are developing strip casters to produce carbon steels (46, 74). Eventually it will be possible to cast carbon steels (74).

Strip casting can achieve considerable energy savings because hot rolling, thus intermediate reheating of the steel, is no longer required. The potential for reduction of the heat requirement for rolling is therefore 100%. Some electricity is still required for various small operations: e.g. ladle handling. We estimate the electricity requirement of strip casters to be 25% of the reference (no data available, own estimate). The SEC based on primary energy units will then be 0.15–0.25 GJ/trs, which means that it is 90–95% lower than that of the SEC of the reference.

Reduction of capital costs compared with a conventional continuous caster with a hot strip mill is estimated to be 55–65% (per tonne of product) (82). O&M costs of a strip caster are estimated to be 50% higher (82). Strip-casting plants are already commercially attractive, with a capacity of 0.5–0.7 million tonnes per year (82).

Strip casting implies the application of a new principle of casting. Major adaptations have to be made to the casting and shaping processes. The product characteristics may also change and open new markets. Strip casting is therefore considered to be a major change.

Spray casting Spray casting involves atomization of the liquid metal and depositing of the formed droplets on a substrate. The droplets are cooled by a gas stream while being deposited. With regard to quality and mechanical-technical properties over strip cast steel, spray cast steel has some advantages. A disadvantage is the low yield. The technique is being applied to some nonflat products (51, 74).

A spray-casting process is being developed by Mannesmann Demag (Germany). A pilot plant has been in operation in Germany, producing flat products of 12–20 mm thickness (51). Other companies that have worked on the development of spray casting are Sandvik Steel (Sweden), Sumitomo Heavy Industries (Japan), and General Electric (United States) (85).

Spray casting for the production of large quantities of steel is still in an early stage of development. Realization of the process for certain specific products, e.g. super alloys, is more probable (74).

As far as the SEC is concerned, no savings over strip casting are expected. No heat is required, as the metal is already molten. We expect a higher electricity demand than for strip casting because electricity is required for spraying the droplets, for maintaining the cooling gas stream, and for some drive power. We estimate the electricity demand to be 50% of the reference. The SEC is then 0.31–0.46 GJ/trs, or about 85% lower than the reference. The overall savings may be smaller when a large amount of steel is lost as a result of a low yield.

Spray casting involves the application of a new principle for casting. Major adaptations to existing process are required. Product characteristics will be

Table 10 Breakdown into energy carriers of the specific energy consumption (SEC) for near-net-shape casting techniques and a reference process

	Final energy (GJ/trs) ^a	Primary energy	
		Low case	High case
		(GJ/trs)	(GJ/trs)
Reference (based on Ref. 8)			
Fuel	1.82	1.82	1.82
Electricity	0.37	0.62	0.93
Steam	-0.15	-0.05	-0.05
SEC		2.38	2.69
Thin slab casting (ISP) (Ref. 83)			
Fuel	0.00	0.00	0.00
Electricity	0.71	1.18	1.76
SEC		1.18	1.76
Thin slab casting with liquid core reduction (own estimate)			
Fuel	0.36	0.36	0.36
Electricity	0.30	0.49	0.74
SEC		0.86	1.10
Strip casting (own estimate)			
Fuel	0.00	0.00	0.00
Electricity	0.09	0.15	0.23
SEC		0.15	0.23
Spray casting (own estimate)			
Fuel	0.00	0.00	0.00
Electricity	0.19	0.31	0.46
SEC		0.31	0.46

^aGigajoule per tonne of rolled steel.

different, affecting the finishing operations. The technique might be used only for speciality steels. Although spray casting can be considered a radical change in casting technique, it is a major change if steel making as a whole is considered.

CONCLUSIONS CONCERNING NEAR-NET-SHAPE CASTING Table 10 gives a breakdown of the SEC of the near-net-shape casting techniques and a reference process, namely continuous casting and a hot strip mill. The maximum saving can be achieved with strip casting. This technology might reduce the SEC of casting and shaping by 90–95%. If this is done, the SEC of an integrated primary steel mill can be reduced from 19 to 16.5 GJ/tcs, or 13%.

Table 11 gives an overview of the other characteristics of the near-net-shape casting techniques. The degree of technical change and the stage of develop-

Table 11 Overview of the characteristics of near-net-shape casting techniques

Technique	Stage of development	Casting thickness (mm)	Capital costs per tonne (index)	Operation and maintenance costs (index)	Degree of technical change
Continuous casting and hot strip mill	State of the art	>150	100	100	—
Thin slab casting	Commercial	40–80	30–55	80–110	Evolutionary
Thin slab casting with liquid core reduction	Commercial/ demonstration	<25	100–120	90–110	Evolutionary
Strip casting	Demonstration/ pilot plant	1–10	55–65	135–165	Major
Spray casting	Pilot plant	5–20	?	?	Major

ment of all techniques are also shown in Figure 17. TSC is already commercial and is therefore placed outside the framework. The techniques with the largest saving potential, strip casting and spray casting, are still not available. The degree to which the techniques differ from the current technique is characterized as major, implying that implementation of these techniques requires major adaptations to the process.

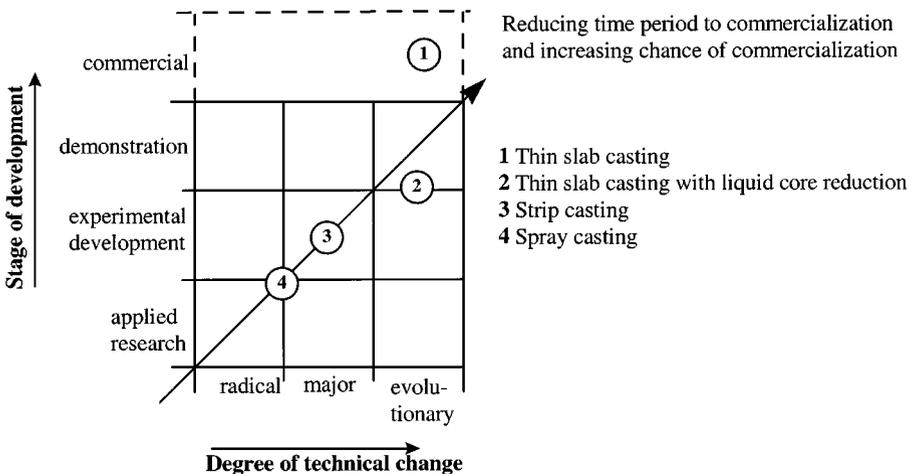


Figure 17 Comparison of near-net-shape casting techniques with regard to the degree of technical change compared with the current technique and the stage of development.

6.3 Scrap-Based Process

In primary steel production, most energy is required to prepare the raw materials and to reduce iron ore. In modern energy-efficient steel mills, the proportion of these processes (including heating and melting of the iron) can be as high as 90% of the total primary energy demand. It is obvious that large energy savings can be achieved if these processes are avoided. Recycling and reprocessing of steel scrap offers this possibility. To adjust shape and properties, however, melting is still required. Several options for melting scrap are available or under development:

1. in a basic oxygen furnace
2. in an electric arc furnace
3. in a scrap melter using both electricity and fossil fuel
4. in an all-fossil fuel melter

The quality of the steel depends largely on the quality of the scrap. Because high-quality scrap is expensive, virgin iron-containing materials can be added to upgrade the quality of the product. Direct reduced iron is frequently used for this purpose. Pig iron can also be used, and recently experiments have been performed in which an EAF is charged with iron carbide (86). Obviously, the use of these virgin materials increases the overall energy consumption, as their production consumes a considerable amount of energy. The quality of scrap can also be upgraded by chemical and mechanical separation processes. The additional energy demand for these processes can be estimated to be 0.5–2 GJ/tonne of scrap (51).

Below we discuss the four options for melting scrap and then assess the potential for energy-efficiency improvement of scrap-based processes.

SCRAP MELTING IN THE BOF Scrap melting in the BOF is common practice. The heat for melting is generated by the oxidation of carbon in the pig iron. Without additional fuel injection, the maximum scrap input is limited to about 25–30% of the charge. A higher scrap ratio can be processed when additional fuel is injected. In fact, Klöckner developed a process for melting a 100% scrap charge in a BOF-like converter. (This process is explored further in the discussion on scrap melting with fossil fuel only.)

ELECTRIC ARC FURNACES Electric arc furnaces (EAFs) can operate on a 100% scrap charge, but they can also process mixed charges including DRI, iron carbide, and cold or hot pig iron. EAF capacity is growing rapidly, from about

18% of the world steel production in 1975 to 27% in 1990 (see Figure 4). EAFs have some advantages over integrated steel mills, e.g. lower capital costs, the possibility of economic operation at low capacity, and smaller environmental impact (86). A drawback is that not all products can be produced. Traditionally, EAFs produce low-quality long products. The low quality of the products is due to the fact that scrap is usually more contaminated than pig iron. Hot rolling mills for flat products can be operated competitively only at large capacities and are therefore usually not installed at EAF facilities. This picture is changing. Near-net-shape casting techniques make the production of flat products from EAF steel competitive with primary steel products. In fact, Nucor in the United States has already entered the flat product market with EAF steel (87).

The principle of EAFs is that steel is melted via electric arcs between cathode and one (for DC) or three (for AC) anodes. The anodes can be placed just above the baths or be submerged in the bath. Oxygen can be injected to promote metallurgical reactions, coal powder can be added to promote slag foaming through CO formation, and oxy-fuel burners may be directed at cold spots.

The major energy input to EAFs is electricity. A reduction of 35% in electricity consumption has been achieved in the past 30 years, as is shown in Figure 3. Even lower electricity consumption levels have been attained by partially replacing electricity by fossil fuel and scrap preheating. At present the most energy-efficient EAF is the finger shaft furnace from VAI and FUCHS (88, 89). The finger shaft furnace makes optimum use of the energy available in the process gases by preheating scrap in a shaft placed above the furnace. Operational results of a 100% scrap-charged finger shaft furnace at Von Roll/SWG in Switzerland show that a primary energy demand of 3.7 GJ/tcs can be achieved, using a 40% efficiency to convert electricity to primary energy carriers (88, 90). Table 12 gives a breakdown of the SEC into energy carriers. Virgin iron materials, like pig iron and DRI, cannot be preheated in the shaft, because they would reoxidize. However, these materials can be charged directly into the furnace. With a 55% DRI and 45% scrap charge, an SEC of 4.6 GJ/tcs (primary energy) has been achieved in a DC finger shaft furnace at HYSLA in Mexico (see Table 12) (90). The higher SEC compared to the 100% scrap-charged furnace is due to the fact that (a) the DRI is not preheated and (b) the DRI has slags, which have to be melted as well. The production of DRI requires about 10 GJ/tonne, which has not been included in this SEC. A third finger shaft furnace at Cockerill Sambre in Belgium is charged with up to 35% hot metal. Because the hot metal does not have to be melted, the electricity requirement can be as low as 230 kWh/tcs; the primary SEC is 2.4 GJ/tcs (88). The losses in modern EAFs are due to cooling water losses, slag heat loss, and chemical and thermal energy loss with the off-gas (91).

Table 12 Breakdown of the specific energy consumption (SEC) of scrap-smelting processes^a

Process and energy carrier (Ref.)	Unit per tcs	Low case (GJ/tcs)	High case (GJ/tcs)
Von Roll Finger shaft furnace (100% scrap) (88)			
Power	315 kWh	1.9	2.8
Oxygen	27.5 Nm ³	0.1	0.1
Natural gas	7 Nm ³	0.2	0.2
Electrode	1.4 kg	0.0	0.0
Carbon powder	15 kg	0.4	0.5
SEC		2.7	3.7
HYLSA Finger shaft furnace (55% DRI; 45% scrap) (89)			
Power	432 kWh	2.6	3.9
Oxygen	31 Nm ³	0.1	0.1
Natural gas	4 Nm ³	0.1	0.1
Electrode	1.2 kg	0.0	0.0
Carbon powder	15 kg	0.4	0.5
SEC		3.3	4.6
Combination shaft furnace (100% scrap) (design value) (51)			
Power	180 kWh	1.1	1.6
Oxygen	65 Nm ³	0.2	0.2
Natural gas	0 Nm ³	0.0	0.0
Electrode	0.8 kg	0.0	0.0
Carbon powder	35 kg	1.0	1.1
SEC		2.3	3.0
K-ES (100% scrap) (92, 93)			
Power	300 kWh	1.8	2.7
Oxygen	50 Nm ³	0.1	0.2
Natural gas	4 Nm ³	0.1	0.1
Electrode	3 kg	0.1	0.1
Carbon powder	22 kg	0.6	0.7
SEC		2.8	3.8
KS (100% scrap) (93)			
Power	0 kWh	0.0	0.0
Oxygen	300 Nm ³	0.7	1.1
Natural gas	1.5 Nm ³	0.0	0.0
Electrode	0 kg	0.0	0.0
Coal	255 kg	7.4	8.2
SEC		8.2	9.3
KVA (100% scrap) (93)			
Power	0 kWh	0.0	0.0
Oxygen	70 Nm ³	0.2	0.3
Natural gas	40 Nm ³	1.3	1.3
Electrode	0 kg	0.0	0.0
FeSi	15 kg	2.2	2.2
SEC		3.6	3.7

^aExpressed in gigajoules (GJ) of primary energy per tonne of crude steel (tcs). The following conversion factors are used: from electricity to primary energy carriers, 0.6/0.4 (low/high); electrode, 30.95 GJ/tonne; oxygen, 280 kWh/tonne; natural gas, 31.65 megajoule (MJ)/Nm³; coal/carbon powder, 29/32 GJ/tonne (low/high); FeSi, 0.14 GJ/kg.

The energy required for heating scrap from 25°C to melting point and subsequent melting is between 1.05 to 1.36 GJ/tonne; the lower limit is for pig iron and the upper limit is for pure iron. However, the theoretically lowest energy consumption for making steel from scrap is low. When we compare the SEC of modern EAFs with the energy required for melting, we can conclude that EAFs are efficient melters. The recovery of the thermal energy from the hot molten steel, on the other hand, is far less efficient. This needs to be improved to reduce the SEC of steel making from scrap still further. We discuss this option in section 6.5.

Large losses occur in the conversion of primary energy to electricity. If the efficiency of electricity generation can be raised from 40% to 60%, the SEC of the finger shaft furnace (100% scrap charge) will reduce from 3.7 to 2.7 GJ/tcs, without improvements in the EAF itself. Melting of scrap with fossil fuel only, thus avoiding the use of only electricity, is discussed later.

Developments in EAF technology are directed toward (a) increasing productivity by decreasing the tap-to-tap time and increasing the capacity and (b) reducing operation costs by reducing power and electrode consumption (51). The double shaft furnace (VAI/FUCHS), which has two separate shaft furnaces that are served by one set of electrodes and one transformer, can achieve a production of 1.2 million tpa, as compared to 0.7–0.8 million tpa for a finger shaft furnace (90). The SEC is on the same order as that for the finger shaft furnace. One step further is the combination shaft furnace (VAI/FUCHS), which has an efficient scrap preheating shaft that can be rotated from one furnace to the other. The scrap is preheated to about 1000°C. From Figure 6 it can be seen that preheating to this temperature requires approximately 0.7 GJ/tonne. Thus, half of the theoretical energy demand for heating and melting is provided in the preheating shaft. According to the developer, this furnace can produce crude steel with a power consumption of 180 kWh/tonne; 65 Nm³ of oxygen; 35 kg of carbon; and 0.8 kg of electrode (51). A SEC of 3.0 GJ/tonne (primary energy; 40% efficiency of electricity generation) results (see Table 12). When electricity is generated with 60% efficiency, the SEC will come down to 2.3 GJ/tonne.

SCRAP MELTING USING BOTH ELECTRICITY AND FOSSIL FUEL Scrap melting using both electricity and fossil fuel is a technique already used in EAFs. The Klöckner electric steel making (K-ES) process was developed to partially replace electricity by coal or coke (92, 93). Enhanced bath stirring is achieved by bottom injection of inert gases. With a coal injection of 22–30 kg/tcs, the electricity consumption is 300 kWh/tcs (92, 93). The SEC on a primary energy basis is 2.8–3.8 GJ/tcs (with an efficiency of electricity generation of 40%) (see Table 12). Three companies (two in Italy and one in Japan) operated the K-ES process in 1993. Mannesmann-Demag has been studying the melting machine,

which also used bottom fuel injection and electricity for scrap melting. A design electricity use of less than 300 kWh/tcs has been reported (51). However, other data are not available.

SCRAP MELTING WITH FOSSIL FUEL ONLY At least two processes that melt scrap with fossil fuel only have been proposed, the Klöckner steel making (KS) process and the Klöckner Voest Alpine (KVA) process (92, 93). The KVA is a continuous scrap-melting process using oxygen and natural gas burners to melt the scrap, applying postcombustion (51). Refining of the molten steel has to be done in a separate reactor. R&D concerning this process stopped because of environmental problems and a high scrap price (L Hofer, Voest Alpine Industrieanlagenbau GmbH, personal communication). The KS process melts scrap using bottom coal injection, oxygen, and postcombustion. After melting, the converter can switch to the refining operation. The SEC of the KVA process is about 3.6 GJ/tcs; the SEC of the KS process is 8.6 GJ/tcs (see Table 12) (93).

The SEC of the KVA process is more or less equal to the most efficient EAF in terms of primary energy consumption. In terms of final energy consumption, a modern EAF requires about 1.4 GJ of electricity to melt scrap, whereas the KVA process requires 3.6 GJ of fossil fuel. The transfer of the heat of hot gases to a bulk of metal is apparently less efficient than heating by electricity. The result is a relatively large volume of waste gas that is produced in the furnace. When electric melting is applied, this volume of gas is produced in a power plant, which is designed to optimally recover the thermal and chemical energy of the gas. Utilization of the waste gas of the melting furnace is hampered by environmental problems caused by impurities in the scrap. To reduce the SEC of fossil fuel-fired melting, R&D should be directed toward improving the heat transfer of the hot gases to the scrap, production of clean waste gas, and optimal use of the thermal and chemical energy of the waste gas.

CONCLUSIONS CONCERNING SCRAP-BASED PROCESSES The most efficient process for melting a 100% scrap charge is the single shaft furnace with a SEC of 3.5 GJ/tcs (assuming a 40% conversion efficiency for power plants). The most efficient EAF process under development is the combination shaft furnace, which has a design SEC of 2.9 (40%) to 2.3 (60%) GJ/tcs. This process uses both fossil fuel and electricity to melt scrap. The most efficient fossil fuel-based process, which is not yet commercial, has a SEC that is comparable to the most efficient EAF in terms of primary energy. To decrease the SEC of fossil fuel-based processes, the large volume of gas needs to be used efficiently.

Although the SEC of the future EAF approaches the minimum energy requirement for heating and melting iron, a further reduction of the SEC may be achieved when ways are found of recovering the thermal energy that is now

lost with the waste gases, the hot metal, and the slag. Heat recovery from the waste gases to preheat scrap is already commercial. Slag waste heat recovery and heat recovery from the molten steel have not yet been demonstrated. We deal with these options in Section 6.4.

The technical changes needed for the implementation of new melting processes are considered to be evolutionary. Many improvements have been realized in the past few decades, but the principle of electric melting has not changed.

6.4 *Steel Making at Lower Temperatures*

The ultimate technique for reducing the need for high temperatures would be steel making at room temperature, without any temperature rise. Since the reduction of iron ore at room temperature is thermodynamically and kinetically unfavorable, such a process is hard to conceive. The various unit operations, however, can be operated at lower temperatures than in present processes, although these temperatures are usually still far above room temperature.

COKE Coke can be produced at a lower temperature (800°C instead of 1100°C) by completing the heating of the coke while it descends into the blast furnace (51). This process has been tested on a small scale by Kobe Steel in Japan. A saving of about 15% on the fuel consumption of coke making can be achieved (51). This process partially integrates coke making and iron making. However, cooling of the hot coke still occurs.

IRON OXIDE Iron oxide can be reduced without melting the iron. In fact, direct reduction (DR) processes do this. DR technology uses a synthesis gas (or solid fuel directly) to reduce the iron oxides at temperatures below the melting point (900°–1000°C, compared with 2000°C at maximum in a blast furnace). Direct reduced iron (DRI) is physically similar to the ore feedstock and contains the minerals that were originally present in the ore. Many DR processes have been developed (94). At present, the MIDREX DR process has the largest share. It is used to produce 65% of the global DRI (95). Only 5% of the iron produced globally is DRI, the remainder being pig iron produced in blast furnaces (33).

The MIDREX process consists of three reactors. In the first reactor, natural gas is preheated by heat exchanging with off-gases. In the second reactor, the preheated natural gas is reformed to a mixture of CO and H₂. The reformed gas is fed to a shaft furnace where iron ore, in the form of pellets or lump ore, is reduced. An improvement to the MIDREX process is the Arex process, which has been applied in Venezuela (94). The improvement lies in the fact that hot DRI in the shaft reactor is used as catalyst for the natural gas reforming. The gas reforming and the iron ore reduction can therefore be performed in the same reactor, resulting in lower investment costs and a lower SEC.

Not only does the MIDREX process (thus, the Arex process) work at a lower temperature than a blast furnace, but coke making and sintering are no longer required. Thus, the high-temperature processes in the raw material processing are eliminated as well. However, one new high-temperature process is added, namely the production of reducing gas. Ore can be fed as lumps and pellets. Pelletizing still requires a high temperature. The DRI is either cooled in the lower part of the reduction furnace, giving off its heat to a cooling gas, or pressed while hot in briquets, which are then cooled. In both cases the heat, amounting to about 0.4 GJ/tonne of DRI, is lost (96).

Two values of the energy requirement for an Arex plant are reported: (a) 8.7 GJ of natural gas and 70 kWh of electricity per tonne of DRI (6) and (b) 9.7 GJ/tonne of DRI (94). We use an average of 9.5 GJ/tonne. (When the electricity demand is converted to primary energy units using a 40% efficiency of public power generation, the energy figures reported by the first source are 9.3 GJ/tonne of DRI on a primary energy basis.) Assuming that all iron ore is charged as pellets, an additional 1.3 GJ of primary energy per tonne is required for pelletizing (assuming a 40% efficiency for electricity generation). The total SEC of 10.8 GJ/tonne is about 25% lower than the SEC of making pig iron in a blast furnace, including coke making and ore agglomeration. However, because of its low carbon content, DRI cannot be charged as the sole raw material in a BOF plant. DRI has to be melted and refined in an EAF plant or can be charged with pig iron to a BOF. The energy use for melting in an EAF plant should therefore be accounted for when different steel-making processes are being compared, resulting in an SEC more or less equal to that of the blast furnace route. A comparison of SECs is presented in the last section of this chapter.

Powder metallurgy can be used to mix iron with other components and to shape iron into specific forms without melting (97, 98). Powder metallurgy is used commercially to make special products; it is characterized by small production volumes and high demands on precise shape and well-defined properties. Worldwide iron powder consumption in 1992 is estimated to have been about 570,000 tonnes (97). Powder metallurgy involves shaping directly from a ferrous (or nonferrous) powder by pressing it into a mold of the desired shape and subsequently heating it in a furnace to bond the fibers together. Thus, shaping does not eliminate the application of high temperatures. The production of powder requires high temperatures as well. More than 50% of the iron powder is made by direct reduction using the Högenäs-process—especially designed for powder production—or the FIOR process (97). Other common methods of producing iron powders start with pure iron, e.g. atomization or the carbonyl reaction (97). As pure iron is required for these processes, they do not offer possibilities for energy-efficiency improvement.

It is unlikely that bulk steel products will ever be made at a commercial rate with this technique. Furthermore, making steel products with powder production requires an amount of energy that is at least equal to the amount required in conventional processes. Powder metallurgy is therefore not considered further in this study.

CONCLUSION CONCERNING STEEL MAKING AT LOWER TEMPERATURES No technology avoids the melting of steel; melting remains necessary for shaping steel. Although direct reduction of iron ore is performed at a temperature that lies far below the melting temperature, the DRI still has to be heated further to be melted. When we compare the DRI-EAF route with the blast furnace BOF route, we see that both routes have comparable SECs. It can be concluded that as long as melting is required to shape the products, a decline in the temperature of iron ore reduction will not result in a significant decrease of the SEC.

6.5 *Waste Heat Recovery at High Temperatures*

The techniques discussed in the previous sections involved a reduction in the application of high temperatures. In this section, we explore techniques under development that can recover heat at high temperatures and make it available as a high-quality energy carrier. First, we discuss techniques that can be applied in the conventional integrated steel mill. Then we look at possible ways of recovering high-temperature heat from streams from future processes.

EXISTING INTEGRATED STEEL PLANTS Table 13 gives an overview of the main hot flows in an integrated steel mill. The sensible heat and the exergy of the flows are based on the reference plant described in Section 4. The total amount of energy lost as a result of heat leaking to the environment is about 5.5 GJ/trs. The exergy is about 3.1 GJ/trs. Table 13 also presents techniques for heat recovery and the present stage of development.

Waste heat recovery is already applied in integrated steel mills for some clean gaseous flows, like exhaust gases from combustion processes. Most process gases, however, contain dust and have to be cleaned before they can be redistributed. The feasibility of heat recovery at higher temperatures depends on the development of a high-temperature gas-cleaning system. Several systems are being developed or are already commercial, e.g. those based on electrostatic precipitation, ceramic filters, and high-efficiency cyclone separators (99).

The main process gases are blast furnace gas, coke oven gas, and basic oxygen furnace gas. Blast furnace gas is usually wet-cleaned before being expanded in top gas recovery turbines. The gas enters the turbine at environmental temperature. To recover the sensible heat of the gas, dry cleaning techniques should be applied. The entry temperature to the expansion turbine can be raised to about 120°C, which increases the power output by 30–35% (51).

Table 13 Waste heat recovery techniques for process gases and solid flows in an integrated steel plant^a

Unit operation	Hot flow (gas/solid)	Sensible heat (GJ/trs)	Exergy (GJ/trs)	Maximum temperature (°C)	Technique	Stage of development
Coke making	Hot cokes (s)	0.24	0.14	1100	Dry coke quenching	Commercial
	Coke oven gas (g)	0.24	0.12	700	Waste heat recovery	Stopped
Sintering	Sinter cooler gas (g)	0.97	0.28	350	Advanced sintering machine or	Commercial
	Sinter exhaust gas (g)	0.23	0.12	350	Emission optimized sintering	Demonstration
Blast furnace (BF)	BF gas (g)	0.82	0.33	500	Top-pressure recovery turbine using dry-cleaning	Commercial
	BF slag (s)	0.39	0.26	1300	Radiant heat boiler	Prototype, R&D stopped since end of 1980s
Basic oxygen furnace (BOF)	BOF gas (g)	0.19	0.12	1200	BOF gas recovery	Commercial
	BOF slag (s)	0.02	0.01	1500	combined boiler/suppressed combustion Radiant heat boiler	Commercial Prototype, R&D stopped since end of 1980s
Casting	Cast steel (s)	1.39	1.06	1600	Radiant heat boilers with heat pipes	Commercial
					Slab cooler boiler	Commercial
Hot strip mill	Hot rolled steel (s)	1.04	0.62	900	Water spraying and heat pumps	Commercial
Total		5.53	3.06			

^aBased on References 51 and 8. GJ/trs, gigajoule per tonne of rolled steel.

Coke oven gas is processed in the by-product plant to recover tars and benzol. A waste heat boiler can be installed at the ammonia incinerator, recovering about 0.01 GJ/trs of steam (51). Some experiments with high-temperature heat recovery from the coke oven gas failed, because of fouling of the heat exchanger (5). The sensible heat of blast furnace gas can be partially recovered using a closed BOF gas system or a combined boiler/suppressed combustion gas recovery system (8, 51). About 0.1 GJ/trs of steam can be produced (8).

The sensible heat from solid streams is yet not being recovered at a large scale, although some techniques are commercial or under development. Dry coke quenching, for instance, is a commercial technique for recovering the sensible heat from hot cokes. The heat is recovered by blowing an inert gas over the incandescent cokes and generating steam by heat exchange with the hot gas (51, 100). However, this technique is expensive (5) and is therefore seldom installed.

Blast furnace slag, and to a lesser extent BOF slag, are also sources of sensible heat losses. In the 1980s several systems were considered for blast

furnace (BF)- and BOF-slag heat recovery. They have never been realized in practice, however, mainly because of the high investment costs (31, 51). In most designs the slag is granulated while heat is recovered in the temperature range of 1500°–1000°C in a steam boiler. Subsequently, heat is recovered by convection to air in the temperature range of 1000°–150°C. One concept used a low-boiling, organic fluid for the recovery of the sensitive heat. Granulation can be achieved by using an air-jet, rotating drums, or a stirrer. These processes can recover about 65–75% of the enthalpy of the slag (31). Developments of these processes have taken place in Japan, Germany, and Great Britain. Using an exergy analysis, Bisio showed that the recovered heat can be best used for preheating combustion air used in the hot blast stoves (31).

Another flow of solid hot material is sinter. The sensible heat of hot sinter is usually partially recovered in the sinter cooler. In our reference plant, for instance, it is assumed that the combustion air is preheated by heat exchanging with sinter cooler air. However, more heat can be recovered by advanced sintering or emission-optimized sintering (51). Advanced sintering involves cooling and sintering in one machine. The waste gas temperature is much higher than in conventional sintering. In emission-optimized sintering, the exhaust of the sinter process is partially recycled to this process. The sinter bed functions as a filter that catches small particles present in the exhaust. The part that is not recycled is water quenched. Emission-optimized sintering is a commercial technology (101).

Finally, there are two flows of hot steel that give off heat to the environment: in the continuous caster and after the hot rolling mill. The initial temperature of the first flow is 1400°–1500°C, and the second flow is about 800°–900°C. The sensible heat of the first flow is usually lost to the environment. This loss can be reduced by charging the slabs to the reheating furnace while they are hot, by transporting the slabs through an insulated tunnel, and by charging them directly to the reheating furnace. This technology—called hot charging—is commercial, and it is implemented at several steel mills around the world. Hot charging is usually not possible for the whole production for logistic reasons, e.g. the capacities of the caster and the rolling mill do not match, or part of the slabs are sold. Furthermore, there is always a temperature drop of about 500°–600°C between casting mold and the entry of the reheating mill.

Can the sensible heat of the cast steel can be recovered? As shown in Figure 6, about 25% of the thermal energy of liquid iron of 1550°C is due to the enthalpy of melting. Recovery of this energy is probably difficult because the surface of the steel solidifies directly after the casting mold. Below 1500°C, the energy that can be recovered declines per tonne of steel approximately linearly with the temperature, namely by 0.7 MJ/°C. Cooling rates are higher at the higher temperatures.

The hot steel after the hot rolling mill is usually cooled by spraying water on the hot steel. The water, with a temperature of 70°–80°C, can be recovered. However, considering that the temperature of the steel is about 800°C, this means a large loss of exergy. The heat can be upgraded by using a heat pump.

Two possible techniques for heat recovery of slabs between the caster and the hot rolling mill have been described in the literature. First, Kashima Steel Works in Japan makes use of radiant heat boilers placed above the slabs (102). A volatile medium, contained in heat pipes placed above the slabs, is vaporized and flows to a steam boiler, where it gives off its latent heat. Per tonne of slabs only 0.005–0.01 GJ of steam is generated. If we assume that heat recovery starts at the beginning of the horizontal part of the caster, at about 800°C, the maximum energy recovery potential is 0.6 GJ/tcs. The first radiant heat boiler was installed as early as 1980. The second possibility, which offers a larger recovery efficiency, is direct transfer of heat to steam in the slab cooler boiler (7). The slabs are conveyed through the boiler while giving off heat, which is used to generate steam of 40 bar and 450°C. With an entry temperature of 900°C and an exit temperature of 300°C, 0.32 GJ of steam per tonne of crude steel can be produced. Investment costs of about US\$25 million for an installation that can process 2.2 million tonnes per year have been reported (7). The payout time is estimated at 6 to 8 years (7). A system like this has been installed at a steel mill in Eisenhüttenstadt (Germany) (JP Torlet, Cockerill Sambre, Research & Development, personal communication).

FUTURE PROCESSES In Section 6.4 we concluded that melting will remain necessary to shape steel. Therefore, at least one heating and cooling step will be necessary. Thus, hot steel, slag, and gas will always be produced. Casting operations in future processes will differ from current continuous casting in several respects. First, strip casting eliminates hot rolling and reheating. Thin cast slabs require hot rolling, but the slabs are charged directly to a soaking furnace, to ensure uniform heat distribution, and then to the rolling mill. The temperature drop between casting and reheating is much smaller. Second, the production speed is much higher, because thinner steel is cast. Third, cooling time is much shorter. Heat recovery might increase the cooling time, because the temperature gradient between the hot steel and the surface of the heat exchanger is smaller than between hot steel and ambient air. (The surface of the heat exchanger will radiate heat also. The emissive power of a surface depends on the temperature to the fourth power. When the temperature of the hot steel is 1800 K and the temperature of the surface of the heat exchanger is 1600 K, 60% of the heat emitted by the steel is returned by the heat exchanger surface. When these temperatures are 1800 K and 1000 K, respectively, only 10% of the heat is returned. The more heat is returned, the longer the cooling time.)

What does this mean for the development needs of recovery techniques? When thin slab casting is applied, heat should be recovered at two places: after the caster and after the rolling mill. Strip casting requires heat recovery at one place. To achieve a high exergetic efficiency, heat should be recovered at high temperatures and in a large temperature range. It could well be that different heat recovery devices should be used for different temperature ranges. In the low-temperature range, an organic medium can be used. By making use of the heat of evaporation, a high heat capacity can be obtained. As described above, these devices are already commercial for conventional continuous casting. At higher temperatures, molten salts may be applied, although their corrosive nature puts high demands on the materials. As far as we know, hardly any R&D is being undertaken concerning the high-temperature heat recovery of cast steel.

CONCLUSION CONCERNING WASTE HEAT RECOVERY AT HIGH TEMPERATURES
Many heat recovery techniques are available, for both gaseous and solid streams. Implementation of these techniques has not been achieved, mainly because of the high investment costs involved. One of the topics of R&D should therefore be to make heat recovery more profitable by recovering a larger part of the heat at higher temperatures. For future steel-making processes, the recovery of the heat of the cast steel over the whole temperature range from 1600°C to environmental temperature is a big challenge. Recovery of heat in the low temperature range can probably be developed first, since this technique is already available for continuous casting. Recovery at higher temperatures still requires much R&D.

6.6 *Conclusions Concerning the Potential of Long-Term Energy-Efficiency Improvement*

This final section is an overview of the expected SECs of future steel-making processes. Furthermore, we discuss to what extent the exergy losses have been reduced and what needs to be done to achieve a further reduction. Finally, we estimate future potential energy consumption from steel making.

Figure 18 gives an overview of four future steel-making routes and the expected SEC, expressed as GJ primary energy per tonne of hot rolled steel.

The first process route is an improved version of the blast furnace route. The SECs are taken from Worrell et al (5). Most of the techniques that have to be applied to achieve this potential have already been demonstrated, and they can be added to the process without major adaptations. Although larger improvements are possible by using newer techniques, it is likely that many integrated steel mills will be adapted in this way, because it involves only evolutionary changes.

The second process route is an advanced primary steel-making route incorporating an efficient smelting reduction process and strip casting. The SEC

is 34% lower than the SEC of the current best-practice integrated mill. All techniques have been proven on a pilot-plant scale and are expected to be commercially available within 15–20 years. The major driving forces are the lower environmental impact and the large reduction in production costs.

The third process route depicted in Figure 18 is based on an EAF combined with direct reduction of iron ore according to the AREX process and strip casting. The AREX process is the most efficient commercial DR process available. To account for energy-efficiency improvement of the AREX process, we assume that the SEC of the future AREX process is 0.5 GJ/tcs lower than that of the current process. This improvement is the same as that which can be achieved in the blast furnace.

The fourth process route is advanced scrap-based electric melting in combination with strip casting. The energy demand for melting scrap is lower than for melting DRI, because DRI contains slags that have to be heated and refined.

All techniques required for these processes can probably be made commercially available within 20 years. Implementation will take considerably longer. In the next century all process routes may be used side by side. The choice of the process will depend on (a) geographical factors, such as the availability of natural gas or cheap electric power; (b) market factors, such as the availability of high-quality scrap and the demand for specific steel products; and (c) the development of the price of steel products. The development of the technologies described is taking place almost completely in the iron and steel sector and depends little on developments in other sectors. However, governmental support is not uncommon. The development of nearly all smelting reduction processes has been supported by the national government.

Is a further reduction in SEC to be expected in the longer term? We discuss the major energy losses of the future processes.

The *advanced primary steel-making route* has a SEC that is about 6 GJ/trs higher than the theoretical value. Where does this energy go, and can this loss be avoided?

First, heating and melting are required in advanced process. The energy of solid streams, namely BF slag, BOF slag, and hot steel, equal to about 2–2.5 GJ/trs, is still lost. Waste heat recovery from the slags is technically possible. No practical means is available for recovering all the heat losses from hot steel. For a further improvement of the SEC, R&D needs to focus on heat recovery from hot steel over the whole temperature range from 1600°C to environmental temperature, including the heat of melting.

Second, the energy losses associated with oxygen production are not accounted for in determining the minimum energy demand. On the basis of primary energy carriers, about 1.2 GJ/trs is required. Oxygen production may become more efficient, for instance by applying membranes. The possibility

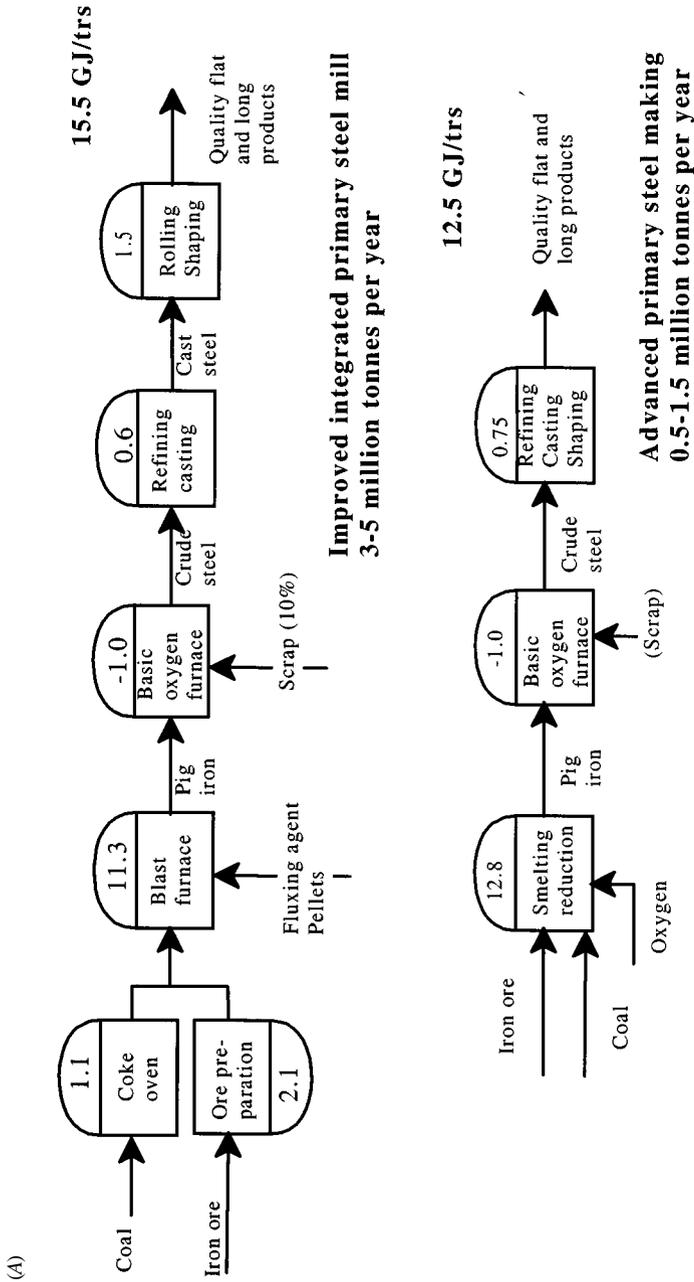


Figure 18 (Continued)

(B)

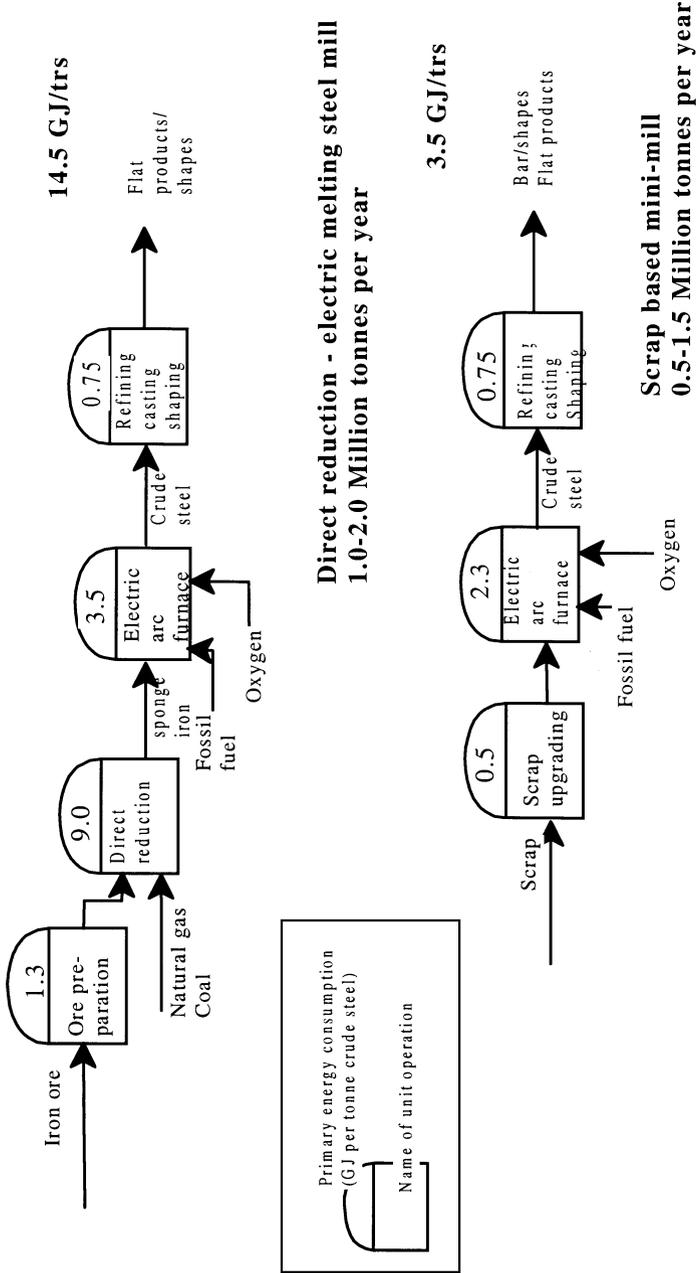


Figure 18 Flow sheets of future steel-making processes. See Figure 5 for a comparison of current steel-making processes. The expected specific energy consumption is also given, expressed in gigajoules (GJ) of primary energy per tonne of hot rolled steel. For an account of these data, see section 5 of this paper.

of using oxygen-enriched air instead of pure oxygen should also be investigated. A negative effect of this option is that the volume of gas that has to be compressed and heated increases.

Third, not all energy of the export gas of the SR process can be recovered. In the model we used, to credit for the export gas it is assumed that in the best case 40% of the enthalpy is lost in the conversion to electricity. In some SR processes, steam is produced by heat exchange with the high-temperature gas. Besides the loss that occurs during this heat exchange, it is assumed that 65% of the enthalpy of the steam is lost in the steam turbine.

Fourth, refining, casting, and shaping require about 0.75 GJ/trs. Refining—removing the last impurities and adjusting the composition to that of the desired steel—requires 0.5 GJ/trs. Strip casting requires 0.15 GJ/trs, and the remainder is for shaping. These values are expressed in primary energy units, whereas the demand is for electricity. The end-use demand may be reduced by several control and optimization measures, e.g. adjustable speed drives.

Finally, the energy input to the SR process is higher than could be expected on the basis of evaluating the reduction of hematite alone. This can be explained by investigating the exergy losses that occur in the SR process itself. Many chemical reactions are responsible for the high-temperature heat: coal gasification, postcombustion, reactions of metals with oxides. The production of heat inherently results in an exergy loss. Exergy is also used to melt other components in the ore, like silicon oxide, which makes up 30–40% of the ore. The resulting molten slag layer is important in the SR process. A better understanding of the function of the slag layer is likely to lead to better control of slag formation, resulting in reduction of the exergy loss.

The SEC of the *future scrap-melting process* is 3.5 GJ/trs, 2.4 GJ/trs higher than the minimum energy demand for melting, achieved in an EAF. Half of this additional energy demand is due to the fact that the input to the EAF is different from that for pure iron. Scrap upgrading and refining both demand 0.5 GJ/trs. Charging the furnace with a raw material with the exact composition of the desired product would diminish this energy demand. The other half of the higher energy demand is due mainly to the conversion of fossil fuels to electricity. Scrap melting in 100% fossil fuel-fired furnaces has been investigated, but so far these furnaces have not been more efficient than EAFs. Efficient utilization of the large volume of waste gas in the furnace itself may be an option to decrease the SEC. The SEC of future EAFs may be further reduced when a way is found to recover the heat of the hot steel. Theoretically, making steel from scrap requires little energy.

In the introduction to this paper, we referred to a study described in a report of the World Energy Council (WEC) that explored future energy demand for steel making under different scenarios (1). Assuming a business-as-usual

development scenario, energy demand might grow from 18.6 EJ in 1990 to 25.4 EJ in 2020. According to the WEC, when advanced technologies are implemented, the demand in 2020 might be 19.5 EJ. Assuming successful development, the techniques described in this paper will probably become commercially available before 2020. The diffusion of these techniques will take place in the decades following market introduction. To make a projection of the potential future primary energy demand for steel making, we make two more assumptions. First, after 2020, steel demand will grow by the same rate as assumed in the WEC study for the period 1990–2020, namely 1.7% a year (1). In addition, growth in developing countries is assumed to be 4% a year. Second, the ratio of primary steel to secondary steel remains at the 1990 level of 70:30 (1). On the basis of these assumptions, world primary energy demand for steel making in 2050 will be 20.8 EJ. Since it can be expected that the SEC will be further reduced by evolutionary changes, and that the use of scrap will increase, future energy demand can be projected to be below the 1990 level. If advanced heat recovery techniques are developed and adopted, an even greater reduction in the energy demand can be achieved.

7. DISCUSSION

In this section we comment on the methodology (10), the choices we made, and the uncertainty of the data.

We opt for a definition of the energy service that allows us to include the recycling of steel scrap. Recycling and reprocessing of scrap has a much lower SEC than does primary steel making and is therefore an important energy-efficiency improvement option. However, it cannot be concluded that all steel should be made according to this process. First, the resources of scrap are not sufficient to meet demand if all steel were to be produced from scrap: World steel demand will grow, it is impossible to collect all steel at the end of its lifetime, and the quality of scrap is not homogeneous. Second, the product mix of a scrap-based mill is different from that of an integrated mill. With the introduction of thin slab casting, this difference has been eliminated to some extent. To circumvent these problems we presented the potentials for energy-efficiency improvement in integrated mills and scrap-based mills separately.

An even broader definition can also be considered, for instance one that includes the production of other materials that can replace steel. We realize that this might result in large energy-efficiency improvements in the long term. Studies that compare the energy requirement for the production of different materials have been published (103, 104). An assessment of the energy-efficiency improvement potential as a result of material substitution requires additional information about and analysis of expected developments in energy efficiency

and in the demand for different products, competition between products, and the emergence of new products.

The selection of techniques was performed on the basis of the results of the exergy analysis. Three groups of techniques can be distinguished. The first group consists of techniques that avoid at least one heating and cooling step. The second group is made up of techniques that reduce the temperature level required in different processes. The third group contains techniques that recover and apply heat at high temperatures.

Noteworthy is the lack of technologies that involve a completely different way of steel making. There do not seem to be any technologies to make steel at lower temperatures. The reduction of iron ore to iron hardly proceeds at low temperatures; the opposite reaction is favored thermodynamically. It can be concluded that the reason no technologies have been found for reducing iron oxide at low temperatures is that no practical ways of achieving this have been discovered.

In theory, the reduction of hematite with carbon is very efficient. The Gibbs free energy of that reaction is 6.8 GJ/tonne of Fe, very close to the Gibbs free energy of decomposition of hematite into the elements, which is 6.6 GJ/tonne of Fe. From an energy point of view there is no reason to look for other reductants. Nevertheless, it has been proposed to use hydrogen (105). One advantage of using hydrogen is that no carbon dioxide is formed. Of course, this is true only when hydrogen is made without the use of fossil fuels. The use of hydrogen does not entail energy saving in itself. To produce 1 tonne of pure iron, about 650 Nm³ of hydrogen is required theoretically. This equals about 6.5 GJ/tonne. This amount is of the same order of magnitude as the minimum energy demand when coke is used.

Now we comment on the accuracy of the input data and on the way in which we had to convert data to make a comparison possible.

We had to rely on figures presented by developers that were based on the results of pilot-plant experiments or were design values. Data on new techniques are rarely supported by other sources of information. Nevertheless, it is possible to make a rough check of the data for SR processes by calculating the expected demand for coal and oxygen.

The main energy input of SR processes is noncoking coal. It can be expected that the coal demand is higher than the coke demand in blast furnaces, as coal still contains 20–30% weight (wt) (38) volatile matter that has to be heated and evaporated. On the other hand, the ash content is a few percent lower (8). The coal demand of SR processes is up to 30% higher than the coal demand of the blast furnace, which is in line with what can be expected.

On the basis of stoichiometric ratios, the oxygen demand (in tonnes) can be determined to be between 90% (for complete conversion to carbon monoxide) and 180% (for complete conversion to carbon dioxide) of the coal demand. SR processes with a high degree of postcombustion have a higher oxygen demand

than processes with a low degree of postcombustion. Oxygen is not only provided externally, it is also generated within the process by the reduction of iron ore. Depending on the composition of the ore, the oxygen released per tonne of Fe is about 300–500 kg. The oxygen released in the prereduction shaft is usually not available for coal combustion. It can be determined that the reported oxygen requirements for second-generation SR processes are well in line with the value that could be expected on the basis of coal requirement, degree of postcombustion, and degree of prereduction. Differences in the reported and the calculated oxygen demand equal 0.05–0.1 GJ/tonne of Fe (primary energy), or less than 1% of the SEC.

On the basis of the foregoing analysis, we can state that the input data on oxygen and coal consumption of all SR processes are consistent with what can be expected from the stoichiometric oxygen requirement and the differences in composition of coking and noncoking coal. We expect that variations in these input data are so small that they do not affect our conclusions with regard to energy-efficiency improvement potential of SR processes.

Data on the investment costs of SR processes are subject to many underlying factors that cannot be assessed easily. For instance, there may be differences resulting from local prices of equipment and the year to which the valuta relate. There seems to be a relation between investment costs and complexity of the process. The CCF, Hlsmelt, and AISI processes are less expensive than the more complex DIOS and COREX processes. We evaluated the effect that varying the investment costs of the SR processes would have on the production costs of hot metal. The production costs with all second-generation SR processes are in all cases lower than in a blast furnace.

For comparison of the SECs, we converted the energy carriers to primary energy carriers using a low and high case and a simple model for in-house electricity production. We showed in Figure 13 that the SECs of SR processes depend strongly on the way in which the export gas is utilized. SR processes with high export gas production are not always more efficient than the blast furnace process. Careful consideration should be given to the matter of what to do with the export gas. We assumed that all export gas is converted to electricity in a combined cycle plant. Other applications of the export gas may be considered as well. For instance, it can be used as fuel for a fuel cell, as reducing gas in DRI processes, or recycled at high temperatures to the melter. These options should be investigated to find the optimum use of energy for the production of iron (and electricity as a by-product).

8. CONCLUSIONS AND RECOMMENDATIONS

In this paper we have analyzed the potential for the improvement of energy efficiency in the iron and steel industry that can be realized in the long term.

We used exergy analysis to show that the main exergy losses in an integrated steel mill are due to the use of high temperatures. On the basis of the results of this analysis, we concluded that long-term energy-efficiency improvement should be directed toward reducing these losses by (a) avoiding intermediate heating and cooling steps; (b) reducing the temperature required in various process steps; and (c) recovering and applying heat at high temperatures.

The focus in this paper was on smelting reduction processes, which avoid coke making and ore agglomeration, and on near-net-shape casting techniques, which avoid or reduce the need for reheating before rolling. By a combination of these techniques, the SEC might be brought down from the current best-practice figure of 19 GJ/trs to 12.5 GJ/tcs, or a reduction of about 35%. The production costs of steel strip from a future integrated mill that uses smelt reduction and strip casting are far below those from a current integrated mill. Both smelting reduction and strip casting are likely to be available within two decades.

Direct reduction has a lower energy requirement than reduction of ore in an SR process, mainly because melting is avoided. However, subsequent melting remains necessary to shape the steel. Because of the low carbon content, DRI has to be melted in an EAF. The SEC of production of steel in the DRI-EAF route is about 2 GJ/trs higher than that of the SR-BOF route.

Electric arc furnaces can make steel from a 100% scrap charge, thus avoiding the need for iron ore reduction. The SEC of steel making of current best-practice EAF mills is about 7 GJ/tcs expressed in primary energy carriers, using a 40% efficiency of electricity generation. This may come down to 3.5 GJ/tcs by the use of more efficient melting furnaces, more efficient casting and shaping techniques, and assuming a 60% efficiency of electricity generation. Steel mills with an EAF have changed considerably over the past decade; they are now competitive with integrated steel mills in the production of flat products, a market that had previously been the monopoly of integrated steel mills. The use of scrap only for the production of steel is not possible, because not enough scrap is available and the quality of scrap is not sufficient to make all steel products. In the future, different routes to produce steel will continue to exist side by side.

For all process routes, a further reduction of up to 2.5 GJ/trs can be achieved when techniques will become available for recovering and applying the high-temperature heat of hot steel and slag. Several concepts of slag heat recovery have been developed. Because of the high investments, none of these concepts has been commercially applied. Heat recovery of the hot steel at temperatures below 800°C is a commercial technology. R&D should be directed at recovering heat at higher temperatures, including recovery of the heat of melting. No such technology is under development.

The selected energy-efficient techniques described in this paper will probably become available before 2020. The diffusion of these techniques will take place in the decades following the market introduction. During this period the techniques will probably be improved, which may result in higher energy efficiency. It can be projected that when all the steel in the world is produced according to the most efficient processes, world energy demand for steel making will stabilize or even decline. In this projection it is assumed that the current ratio of primary to secondary steel making will still be applicable and that world steel production will grow by 1.7% a year on average. In addition, growth in developing countries is assumed to be 4% a year. Further reductions in energy demand can be achieved when advanced heat recovery techniques are developed and adopted and when the use of scrap is increased.

New techniques are being developed within the iron and steel industry itself. However, governmental support is not uncommon. Nearly all smelting reduction processes are being developed with a form of financial support from the government. The main driver for the development of new techniques is a reduction in production costs. Improvement in energy efficiency can contribute to this. The role of the government in improving energy efficiency in the iron and steel industry is still limited. Several areas may be the subject of governmental policy:

1. Financial support for the development of energy-efficient technologies;
2. Encouraging iron and steel companies to implement the most efficient techniques, e.g. through voluntary agreements;
3. Providing an efficient and effective scrap recycling system and stimulating the maximum use of scrap by iron and steel companies;
4. Encouraging research to further improve energy efficiency, e.g. by developing techniques to recover and apply high-temperature heat and processes to make steel directly from iron ore.

ACKNOWLEDGMENTS

We would like to thank K Meijer (Hoogovens Staal), L Hofer (VAI), E Nieuwlaar, and WC Turkenburg (Utrecht University) for providing information, suggestions, and comments on this study. The exergy analysis is based on earlier work by T Ros. We thank the Netherlands Organization for Scientific Research for its financial support.

Visit the *Annual Reviews* home page at
<http://www.AnnualReviews.org>

Literature Cited

1. WEC. 1995. *Energy Efficiency Improvement Utilizing High Technology—An Assessment of Energy Use in Industry and Buildings*. London: World Energy Council
2. Int. Iron Steel Inst. 1992. *Steel Statistics Yearbook 1992*. Brussels: IISI
3. Int. Iron Steel Inst. 1996. *Statistics on Energy in the Steel Industry (1996 Update)*. Brussels: IISI
4. Tilton JE. 1990. World metal demand, ed. E Landsberg, JE Tilton. Washington, DC: Resources of the Future
5. Worrell E, Beer J de, Blok K. 1993. Energy conservation in the iron and steel industry. In *Energy Efficiency in Process Technology*, pp. 91–100. Amsterdam: Elsevier
6. Faure H. 1993. *The new technologies for iron and steelmaking*. Presented at 27th IISI Annual Meeting and Conference, Paris
7. Maier W, Angerer G. 1986. *Rational Energy Use by New Technologies (Rationelle Energieverwendung durch neue Technologien)*. Köln: Verlag TÜV Rheinland GmbH (In German)
8. Int. Iron Steel Inst. 1982. *Energy and the Steel Industry*. Brussels: IISI
9. Chatterjee A. 1996. *Beyond the Blast Furnace*. Boca Raton, FL: CRC
10. Beer J de, Worrell E, Blok K. 1997. Long-term energy-efficiency improvements in the paper and board industry. *Energy—Int. J.* 23:21–42
11. Ottow M, Neiler H, Wessiepe K. 1994. *Iron, Ullmann's Encyclopedia of Industrial Chemistry A*, 14:461–590. Weinheim: Verlag Chemie
12. Juleff G. 1996. An ancient wind-powered iron smelting technology in Sri Lanka. *Nature* 379:60–63
13. Jeans JS. 1882. On the consumption and economy of fuel in the iron and steel manufacture. *J. Iron Steel Inst.* 3:128–79
14. Hammersley G. 1973. The charcoal iron industry and its fuel. *Econ. Hist. Rev.*, pp. 593–613
- 14a. Rossillo-Calle F, Rezende MAA de, Furtado P, Hall DO. 1996. *The Charcoal Dilemma*. London: Intermediate Technology
15. Hyde CK. 1977. *Technological Change and the British Iron Industry, 1700–1870*. Princeton: Princeton Univ. Press
16. Heal DW. 1975. Modern perspectives on the history of fuel economy in the iron and steel industry. *Ironmaking Steelmaking Q.* 4:222–27
17. Kudrin VA. 1985. *Steel Making*. Moscow: Mir
18. Rosenberg N. 1982. *Inside the Black Box*. Cambridge, UK: Cambridge Univ. Press
19. Siemens CW. 1873. On the manufacture of iron and steel by direct process. *J. Iron Steel Inst.* 5:37–91
20. Worrell E. 1995. Advanced technologies and energy efficiency in the iron and steel industry in China. *Energy for Sustainable Devel.* 11:27–40
21. Goldemberg J, Johansson TB, Reddy AKN, Williams RH. 1988. *Energy for a Sustainable World*. New York: Wiley
22. Worrell E. 1994. *Potentials for improved use of industrial energy and materials*. PhD thesis. Utrecht Univ., Utrecht
23. Considine I, Maxwell D. 1974. *Chemical and Process Technology Encyclopedia*. New York: McGraw-Hill
24. Porterfield WW. 1984. *Inorganic Chemistry*. Reading, UK: Addison-Wesley
25. Tierney B, Linehan P. 1994. *Ullmann's Encyclopedia of Industrial Chemistry*. Weinheim: VHC
26. Weast RC. 1983. *CRC Handbook of Chemistry and Physics*. Boca Raton, FL: CRC. 64th ed.
27. Szargut J, Morris DR, Steward FR. 1988. *Exergy Analysis of Thermal, Chemical, and Metallurgical Processes*. New York: Hemisphere
28. Bisio G, Poggi S. 1991. *The possible utilization of the thermal energy discharged from sintering plants of steel works*. Presented at 26th Intersoc. Energy Conversion Eng. Conf., New York
29. Bisio G, Poggi S. 1990. *Efficient energy resource use in the steel industry with particular reference to Italy*. Presented at 25th Intersoc. Energy Conversion Eng. Conf., New York
30. Bisio G. 1993. Exergy method for efficient energy resource use in the steel industry. *Energy—Int. J.* 18:971–85
31. Bisio G. 1997. Energy recovery from molten slag and exploitation of the recovered energy. *Energy—Int. J.* 22:501–9
32. Stepanov VS. 1993. *Analysis of Energy Efficiency of Industrial Processes*. Berlin: Springer-Verlag
33. Int. Iron Steel Inst. 1996. *Steel Statistics Yearbook 1995*. Brussels: IISI
34. Szargut J, Morris DR. 1987. Cumulative exergy consumption and cumulative degree of perfection of chemical processes. *Energy Res.* 11:245–61
35. Ghamarin A, Cambel AB. 1982. Exergy

- analysis of illinois no. 6 coal. *Energy—Int. J.* 7:483–88
36. Spielmann PE. 1924. *The Constituents of Coal Tar*. London: Longmans Green
 37. Nieuwlaar E. 1996. *Enerpack 5*. Utrecht: Utrecht University, Dept. of Science, Technology and Society
 38. Hoffman JP. 1992. Presidential address: oxygen-coal in-bath smelting reduction—a future process for the production of iron and stainless steel? *J. South African Inst. Mining and Metallurgy* 92:253–73
 39. Hoogovens. 1995. *Cyclone Converter Furnace—Leaflets*. IJmuiden, Netherlands: Hoogovens Staal
 40. Innes JA. 1995. *From direct reduction to direct smelting—options for an expanding China*. Presented at Pacific Economic Co-operation Council Minerals and Energy Forum, Beijing
 41. Gool W van. 1986. *Poly-energy Handbook (Poly-energie zakboekje)*. The Hague: PBNA
 42. Int. Iron Steel Inst. 1990. *Statistics on Energy in the Steel Industry (1990 Update)*. Brussels: IISI
 43. Hendriks C. 1994. *Carbon dioxide removal from coal-fired power plants*. PhD thesis. Utrecht University, Utrecht
 44. Takano H, Kitauchi Y, Hiura H. 1988. *Design for the 145MW blast furnace gas firing gas turbine combined cycle plant*. Presented at Gas Turbine and Aeroengine Congress and Exposition, Amsterdam, June 5–9
 45. Deleted in proof
 46. Fruehan RJ. 1994. Effect of emerging technologies on the competitiveness in the steel industry. *Iron Steelmaker* 21:17–21
 47. Bosley JJ, Clark JP, Dancy TE, Fruehan RJ, McIntyre EH. 1987. *Techno-economic Assessment of Electric Steelmaking Through the Year 2000*. Pittsburgh: Cent. Metals Production
 48. IEA. 1993. *Energy Prices and Taxes*. Paris: IEA
 49. Dijk KM van, Dijk R van, Eekhout VJL van, Hulst H van, Schipper W, et al. 1994. *Methanol from Natural Gas*. Delft: Delft Univ. Technol.
 50. Weston TR, Thompson MW. 1996. *The Romelt Process—Applications in the 21st Century Steel Industry*. Fairfax, VA: ICF Kaiser
 51. Stelco. 1993. *Present and Future Use of Energy in the Canadian Steel Industry*. Ottawa: Efficiency and Alternative Energy Technol. Branch, CANMET, Energy, Mines and Resources Canada
 52. COREX. 1996. *COREX—Revolution in Ironmaking*. Linz: Voest-Alpine Industrieanlagenbau GmbH
 53. Cheeley RB, Eichberger E, Bolkart A. 1996. COREX/MIDREX on the way to commercialization. *Direct from Midrex*, pp. 8–9
 54. Puehringer O, Wiesinger H, Havenga BHP, Hauk R, Kepplinger WL, et al. 1991. Practical experience with the corex-process and its potential for development (Betriebs Erfahrungen mit dem Corex-Verfahren und dessen Entwicklungspotential). *Eisen Stahl* 111:37–44 (In German)
 55. Eberle A, Schiffer W, Siuka S. 1996. *Start-Up and First Operational Results of the COREX Plant C-2000 at POSCO's Pohang Works*. Linz: VOEST-ALPINE Industrieanlagenbau GmbH
 56. Millbank P. 1995. Direct route to iron gathers momentum. *Metal Bulletin Monthly (Metals Technology Supplement)* (April):21, 24–25
 57. Prideaux RN. 1996. *The Hismelt process: premium grade metallics for Asia*. Presented at SEAIISI 25th Anniversary Conf., Bangkok
 58. Cusack BL, Wingrove GS, Hardie GJ. 1995. Initial operation of the Hismelt research and development facility. *Iron Steelmaker* 2:13–20
 59. Kreulitsch H, Egger W, Wiesinger H, Eberle A. 1993. Iron and steelmaking of the future. *Steel Times* 221:217–20
 60. Furukawa T. 1994. 5,000 daily tons of direct ion-ore smelting by 2000. *New Steel* 10:36–38
 61. Iron & Steelmaker. 1996. DIOS pilot plant operation ends, process awaits commercialization. *Iron Steelmaker* 23:9
 62. Faure HA. 1993. Development, state of the art and future aspects of steelmaking. *Metallurgical Plant and Technol. Int.* 16:32–41
 63. Langen J van, Meijer K, Corbett M, Margari G. 1992. The cyclone converter furnace. *Journées Siderurgiques ATS 1992* (Paris)
 64. Soldaat A. 1997. Twelve projects against CO₂-emission receive 280 million Dutch guilders (Twaalf projecten tegen CO₂-uitstoot krijgen 280 miljoen gulden). *Energie—en Milieuspectrum*, pp. 28–29 (In Dutch)
 65. Meijer HKA, Teerhuis CP, Flierman GA, Boom R. 1995. The cyclone converter furnace—recent developments. *La revue de Metallurgie—ATS-JS*, pp. 42–43
 66. Badra C. 1995. *Industry, Trade and Technology Review*. Washington, DC: Office

- of Industries, US Int. Trade Comm.
67. Farley JM, Koros PJ. 1992. Progress towards direct steelmaking in the USA. *Steel Times Int.*, pp. 22–23
 68. Worrell E, Beer J de. 1991. *Gross Energy Requirements—Partial Report on Steel (Energiekentallen—Deelrapport Staal)*. Amersfoort: Castro Consulting Engineer (In Dutch)
 69. Weston TR, Thompson MW. 1996. Bright future seen for the Romelt process. *MBM Metals Technol. Suppl.* 308:6–9
 70. Lassat de Pressigny Y de. 1993. Jupiter—A virgin iron production process for scarp based steel making. *Steel Times* 221: 513
 71. Yemin L, Zhihong X. 1996. *Smelting Reduction Process for Iron and Chemicals*. Peking: Laboratory of Computer Chemistry, Inst. Chemical Metallurgy, Chinese Academy for Sciences
 72. Technisch Weekblad. 1997. Thyssen makes considerable investments in thin slab casting (Thyssen investeert fors in dunne-plakgieten). *Technisch Weekblad*, p. 11
 73. VAI. 1994. Beam Blank Minimill for Stahlwerk Thueringen. *Strand News* 8:6–7
 74. Parodi GG. 1993. *Near net shape casting of flat products*. Presented at 27th IISI Annu. Meet. Conf., Paris
 75. Flemming G, Kappes P, Rohde W, Vogtmann L. 1988. Rolling of continuously cast strips and the technical consequences with respect to the design of hot strip production plants. *MPT* 11:16–35
 76. Szekely J, Trapaga G. 1994. Some perspectives on new steelmaking technologies. *MPT* 17:30–47
 77. Advanced Materials & Processes. 1996. Today's steel technology. *Advanced Materials Processes* 149:32–35
 78. Stam B. 1997. Hoogovens considers thin slab casting (Hoogovens overweegt dunne-plakgieten). *Technisch Weekblad* 11:1
 79. Trouw. 1997. Hoogovens considers combined casting-milling (Hoogovens denkt aan gietwalsen). *Trouw*, March 7, 1997 (In Dutch)
 80. Rohde W, Flemming G. 1995. Stand, Leistungsvermögen und Weiterentwicklung der CSP-Technologie. *Stahl Eisen* 115:89–100
 81. SMS. 1995. *CSP Technology Revolutionizing Hot Strip Production—Leaflet*. Düsseldorf: SMS Schloemann-Siemag
 82. Hendricks C. 1995. Strip casting—a revolution in the steel industry? *Metallurgical Plant Technol.* 18:42–49
 83. Tomassetti G. 1995. The steel industry in Italy: thin slab casting in Cremona. In *Energy Efficiency Utilising High Technology*, Appendix. London: World Energy Council
 84. Kruger D. 1995. CPR—A combined casting/rolling process for producing steel strip. *Iron and Steel Engineer* 72:31–36
 85. Intermediar. 1990. Revolution in steel production (*Revolutie in staalproductie*). *Intermediar* 26:41
 86. Smith T. 1995. European electric steel congress attracts 1000 delegates. *Steel Times* 223:272–73, 278
 87. Mott R, Chase D, Hofmann F, Küper FJ. 1994. *CSP plant Nucor Steel, Hickman: the outstanding performance and the expansion to two strands*. Presented at 2nd European Conference on Continuous Casting, Düsseldorf, June 20–22
 88. Hofer L. 1997. *Electric Steelmaking with FUCHS Shaft Furnace Technology*. Linz: VAI
 89. Hofer L. 1996. Steel production technologies—Fuchs and VAI have joined forces. *Steel Standard—VAI Steelmaking Technol.* 7:2–3
 90. Rong SW, Jian L, Fuchs G. 1996. Compact mill concepts in electric furnace steel making. *Steel Standard—VAI Steelmaking Technol.* 7:9–14
 91. McIntyre EH, Landry ER. 1992. *Electric Arc Furnace Efficiency*. Pittsburgh: EPRI Cent. Materials Production
 92. Teoh LL. 1989. Electric arc furnace technology: recent developments and future trends. *Ironmaking Steelmaking* 16:303–13
 93. Patuzzi A, Antlinger K, Grabner H, Krieger W. 1990. *Comparative study of modern scrap melting processes*. Presented at 6th Int. Iron and Steel Congress, Nagoya, Japan
 94. Zervas T, McMullam JT, Williams BC. 1996. Gas-based direct reduction processes for iron and steel production. *Int. J. Energy Res.* 20:157–85
 95. Midrex. 1996. *1995 World Direct Reduction Statistics*. Charlotte, NC: Midrex Direct Reduction Corp.
 96. Midrex. 1991. *Midrex—Economical Ironmaking Alternatives for Modern Steelmaking*. Charlotte, NC: Midrex Direct Reduction Corp.
 97. Thümmler, Oberacker R. 1993. *Introduction to Powder Metallurgy*. The Institute of Materials Series on Powder Metallurgy. London: Inst. Materials
 98. Salak D. 1995. *Ferrous Powder Metallurgy*. Cambridge: Cambridge Int. Sci.
 99. Ree R van, Oudhuis ABJ, Faaij A,

- Curvers APWM. 1995. *Energy from Biomass: An Assessment of Two Promising Systems for Energy Production—Report 3.3: Modelling of a BIG/CC System with Aspen+*. Petten, Netherlands: Netherlands Energy Research Foundation
100. Dungs H, Tschirner U. 1994. Energy and material conversion in coke dry quenching plants as found in existing facilities. *Coke Making Int.* 6:19–29
101. Hoogovens. 1996. *Environmental Report 1995 (Milieujaarverslag 1995)*. IJmuiden, Netherlands: Hoogovens Staal (In Dutch)
102. Yoshida K, Kobayashi T, Tanaka M, Watanabe T. 1982. *Energy savings in continuous casting*. 4th Int. Iron and Steel Congress, London, May
103. Boustead I, Hancock GF. 1979. *Handbook of Industrial Analysis*. New York: Wiley
104. Worrell E, Heijningen RJJ van, Castro JFM de, Hazewinkel JHO, Beer JG de, et al. 1994. New gross energy-requirement figures for materials production. *Energy—Int. J.* 19:27–40
105. Gretz J, Korf W, Lyons R. 1991. Hydrogen in the steel industry. *Int. J. Hydrogen Energy* 16:691–93
106. Gielen DJ, Dril AWN van. 1997. *The Basic Metal Industry and Its Energy Use*. Petten, Netherlands: ECN
107. Netherlands Ministry of Economic Affairs. 1994. *From World Market to End User (Van Wereldmarkt tot Eindverbruiker)*. The Hague: Ministry of Economic Affairs (In Dutch)
108. Bock M, Boyd GA, Karlson SH, Ross M. 1994. Best practice electricity use in steel minimills. *Iron Steelmaker* 21(5):63–66
109. Gomez M, Gayle JG, Taylor AR. 1965. *Heat Content on Specific Heat of Coals and Related Products, US Bureau of Mines Report of Investigation 66076*, Washington, DC